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THE UNIVERSITY OF ALBERTA

THE EFFECT OF SORPTION ON STRENGTH OF COAL AND ITS INFLUENCE
ON COAL OUTBURSTS

by

Yusuf Ates



A THESIS

SUBMITTED TO THE FACULTY OF GRADUATE STUDIES AND RESEARCH
IN PARTIAL FULFILMENT OF THE REQUIREMENTS FOR THE DEGREE
OF Master of Science

IN

MINING ENGINEERING

DEPARTMENT OF MINING METALLURGICAL AND PETROLEUM ENGINEERING

EDMONTON, ALBERTA

SPRING, 1987

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ISBN 0-315-37730-5

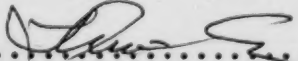
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The undersigned certify that they have read, and recommend to the Faculty of Graduate Studies and Research, for acceptance, a thesis entitled THE EFFECT OF SORPTION ON STRENGTH OF COAL AND ITS INFLUENCE ON COAL OUTBURSTS submitted by Yusuf Ates in partial fulfilment of the requirements for the degree of Master of Science in MINING ENGINEERING.

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ABSTRACT

A sudden violent failure of coal accompanied by the release of large quantities of gas (usually methane and/or carbon dioxide) is known as an "outburst" and represents a major coal mining hazard. Outburst occurrences are affected by many factors such as coal rank, porosity, permeability, strength, gas content and sorption, the field stress and geological structure. These factors are reviewed.

One important parameter affecting outbursts is the coal strength. Gases such as carbon dioxide sorb onto coal surfaces and will reduce the free surface energy and, theoretically at least, will reduce the coal strength by a factor over and above that to be expected from the laws of effective stress due to gas pressure alone. It is, however, unclear from the previous studies whether the magnitude of this postulated strength reduction due to gas sorption is significant.

In this study the tensile strength of coal specimens saturated with both non-sorbing gas (Helium) and sorbing gas (Carbon dioxide) has been measured with the objective of determining if there is a significant reduction in coal strength attributable to gas sorption. A special apparatus was designed and built to allow Brazilian tensile strength tests to be carried out in vacuum and under gas pressures from atmospheric to 3.45 Mpa (500 psi).

Preliminary tests indicated a significant (14%) reduction in strength between helium and carbon dioxide

saturated specimens at 3.45 Mpa gas pressure. It was, however, realised that due to the test procedure this strength reduction might be caused by effects other than sorption, such as specimen damage during gas saturation. The test procedure was therefore changed to ensure that at no stage during saturation process could the gas pressure gradient inside the specimen exceed the strength of coal.

Results using the new procedure clearly showed that the previously observed difference was eliminated. It was therefore concluded that, for this particular coal, gas sorption produced no significant reduction in coal strength for a gas pressure range up to 3.45 Mpa.

ACKNOWLEDGEMENTS

The author wishes to express his gratitude and appreciation for the guidance, encouragement and understanding of Dr. Ken Barron, Professor of Mining Engineering, under whose supervision this investigation was conducted.

Acknowledgements are also made to Professor W. Griffin for his suggestions on the random testing of the samples; to Professor T. Patching for his continuous interest in the project and for numerous beneficial discussions; and to Professor A. J. Hargraves for his valuable suggestions.

The help and cooperation of Mr. D. Booth is appreciated. The author wishes to thank Mr. B. Konzuk, Mr. J. Gibeau and Mr. B. Mohamedbhai for their assistance in various phases of the project. He also thanks the managements of Highvale mine and Metropolitan colliery for providing the coal used in the study.

The author is grateful to NSERC and the Central Research Fund of the University of Alberta for their financial support.

The author wishes to thank his wife, Nezihe, and his children, Senem and Murat, for their forbearance during the many months in which he placed this study above his responsibilities to them.

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NOMENCLATURE

A	Crossectional area
a	Spacing between neighbouring atomic planes
A _m	Average area covered by a single molecule of adsorbate
c	Half the crack length
C	Concentration
C _o	Equilibrated concentration
D	Diameter of the disc sample
d	Diffusion coefficient
daf	Dry ash free
dmmf	Dry mineral matter free
dof	Degree of freedom
E	Young's modulus
E _n	Entropy of the system
g	Grams
h	Depth of cover
i	intercept
k	Fracture permeability
°K	Degrees Kelvin
k _a	Apparent permeability
L	Length of the specimen
M	Molecular weight of gas
N _a	Avagadro's number, 6.022×10^{23}
n	Number of moles
P	Pressure
P _m	Mean pressure

P_o	Liquefaction pressure
r	Radius
R	The gas constant
Q	Rate of flow
S	Internal surface area
S	Total surface area
STP	Standard temperature and pressure, 0 °C and 1 atmosphere
sl	Slope
T	Temperature
t	Thickness
V	Volume
V_m	Volume of adsorbate forming monolayer
V_d	Sorption capacity of dry coal
V_h	Sorption capacity of humid coal
V_m	Volume of adsorbate forming monolayer
w	Moisture content
a_a, a_d, β, λ	Proportionality constants
ξ_o	Eccentricity of the ellipse
Γ	Number of moles of gas adsorbed per unit area
γ	Surface energy
γ_g	Surface energy with adsorption
ζ	Gas viscosity
η	Unit weight of rock
θ	Fraction of the total surface covered
κ	The principal stress ratio, σ_3/σ_1
μ	Population mean

ν	Poisson's ratio
π	3.1415...
σ	Normal stress
σ_a	Applied stress
σ_0	Molecular cohesive strength
σ_v	Vertical stress
σ_t	Tensile strength
σ_1	Major principal stress
σ_2	Intermediate principal stress
σ_3	Minor principal stress
σ'	Effective normal stress
τ_f	Shear stress at failure
x	Amount of adsorption
ϕ	Internal friction angle
ρ	Cohesion of the material
ΔG	Change in the free energy of the system
ΔH	Change in the heat content of the system
ΔE_n	Change in the entropy of the system
$\bar{x}$	Sample mean
s	Sample standard deviation

1. INTRODUCTION

1.1 COAL OUTBURSTS AND THEIR HAZARDS

An outburst can be defined as a sudden, rapid projection of coal and gas, away from the coal face (usually freshly exposed) during mining operations.

A large quantity of gas (methane, carbon dioxide, or a mixture thereof) is released. As much as 800,000 cubic metres of gas release has been reported in the literature (Kidyibinsky 1980). Coal is ejected away from the face in powdery form during the process. This fine powder is called "mad flour" in France, and "bulldust" in Australia (Jackson, 1984). The amount of coal ejected ranges from a few tonnes to 5000 tonnes (Kidyibinski 1980). A cavity, with the gas escape channel on the top, is usually produced in the face of the opening as result of coal ejection.

The whole process of an outburst can occur within a few seconds, with little or no previous warning. Violent noises may precede or accompany the burst. These noises are described in various ways: "...ranging from a crescendo of knockings to sound of falling timber, to roar of wind" (Hargraves 1983). "In West Wales, these noises are known as 'pouncing' and have been compared to those from a two stroke engine" (Jackson, 1984). Rice (1918) made comparison to a small scale earthquake in order to explain the sudden and violent nature of an outburst which occurred in 1904, at Carbonado Colliery No. 1 mine of British Columbia.

Although the intensity of the problem varies from country to country, all the coal mining countries have suffered from this phenomenon. Therefore the subject of outbursts is of almost universal concern.

In Canada, outbursts have been a serious problem in the Crowsnest Pass and Cascade coal areas. For example, in the Canmore area some of the downward extension of seams with good quality coal had to be abandoned due to outburst hazards (Patching, 1985). In the Crowsnest Pass area, mines had to be closed due, mainly, to outburst problems (Grant, 1985). In the Nova Scotia coal fields, especially in No.26 mine, outbursts were reported since 1977 (Jackson 1984).

The worst case reported in Canada (also among the worst cases reported in the world) was the outburst that occurred in 1904 at No.1 mine of Carbonado Colliery, British Columbia. About 3500 tonnes of coal were ejected during driving of a drift. The amount of gas released in the first twenty minutes after the outburst was estimated as between 56,000 and 140,000 cubic metres. The drift was almost closed with powdery coal for 120 m from the face. Fourteen workers lost their lives (Rice, 1918).

The immediate dangers of outbursts can be summarized as:

1. The shock and burial effect of ejected material : The displaced material can bury the personnel in the immediate area of the face.
2. The possibility of suffocation by dust and oxygen

deficient mine air.

3. The possibility of explosion caused by methane as a result of damaged electrical equipment or sparks of a mechanical nature.
4. The ejection of the coal that provides roof support, thus permitting roof falls in the vicinity of the outburst area.

Apart from their violent nature, outbursts also cause production delays, costly repairs, and even mine closures.

Although the main parameters influencing the occurrence of outbursts are identified in broad terms, the mechanism of outbursts is not fully understood.

Therefore, the problem continues. Researchers have not yet been able to find a procedure or test by which the occurrence of an outburst can be predicted. The problem is basically seen as a stress-gas phenomenon; consequently efforts towards the reduction of risk are concentrated towards de-stressing and de-gassification of the coal seams in the areas similar to those where outbursts are known to have occurred in the past.

1.2 FACTORS AFFECTING COAL OUTBURSTS

The following factors increase the susceptibility of coal to outbursts (as listed by Hargraves, 1983 - with minor changes-):

1. Depth of cover.
2. The presence of gas in the seam, in suitable quantity.

3. Geological structure; faulting, folding.
4. High coal rank, high coal friability.
5. Intrusions of seam by dykes.
6. Steep dip of coal seam.
7. Low moisture content of coal.
8. Low coal permeability.
9. Low strength of coal.
10. A percussive effect due to blasting or other shocks in the vicinity.

It should be noted that some of these factors are interrelated. For example, high stresses will result lower coal permeability. High rank coals usually have low moisture content, and the coal is more friable near the geological disturbances, etc. Therefore, it is possible to consider the effects of the above listed factors, in general, as:

1. The state of stress.
2. The strength of coal.
3. The effect of gas, and
4. The coal permeability.

Of these factors, the stress-strength relationships are well studied and can be expressed with an appropriate failure criterion. The reduction in permeability with high stresses is also known to a certain extent. The effect of gas pressure on failure can be formulated by using the law of effective stress. However, there are some indications, from a theoretical basis and on the basis of some indirect experimental studies, that the presence of gas in a coal

seam also effects the strength of coal in a manner over and above that which would be expected from the law of effective stress.

1.3 COAL STRENGTH AND GAS

The influence of gas on the strength of coal was noted a long time ago. Ettinger and Lamba (1957) mentioned that miners observed a weakness in coal when gas was present, and that by preventing air reaching a dead-end, the coal was easier to dig thus resulting in higher productivities.

There have been a limited number of studies aimed at measuring the strength of coal in gas environments (of particular interest are CO_2 and CH_4). Because of the difficulties in preparation of suitable coal specimens and the requirement of special testing equipment, all of these studies used an indirect strength measurement method'. The results obtained are not conclusive and are sometimes contradictory.

It is believed in general that the effect (if any) of gas on the strength of coal (other than pressure effect) is due to adsorption of the gas molecules on coal surfaces.

' . Example: Measurement of the quantity of certain size fraction of coal obtained as a result of dropping a weight on coal in the gas medium - Ettinger and Lamba 1957.

1.4 BROAD OBJECTIVE OF THE STUDY

The objective of this study, therefore, was to investigate the effect of gas media on the strength of coal. To achieve this objective the tensile strength of coal was measured in two different gas environments: Carbon dioxide -which adsorbs on to coal surfaces- and Helium -which is inert, or has negligible sorptive capacity for coal. Comparison of these measurements at different pressure levels was expected to reveal the effect (if any) of sorptive gas on the strength of coal.

1.5 ORGANISATION OF THE THESIS

The subject of investigation is of interest in the understanding of the mechanism(s) of coal outbursts. Therefore the following chapter of this thesis is devoted to a review of the coal outbursts. The subjects of coal-gas relationships, gas sorption on coal, and the mechanism of coal outbursts are reviewed in this chapter (Chapter 2).

A detailed description of the problem, along with the technique adopted for the solution is given in Chapter 3.

Chapter 4 discusses the previous work done in the area of interest. Theoretical strength of brittle materials, and the adsorption - surface energy relations are also covered in Chapter 4, with a purpose of relating the effect of adsorption to the strength of coal. An illustrative example, on the reduction of surface energy with adsorption and thus the reduction in strength of coal, can be found at the end

of this chapter.

Chapter 5 gives the the coal types that were tested: their proximate analyses, ΔP test results with a related discussion.

Chapter 6 discusses the equipment and the testing methods employed.

The results of preliminary tests, their interpretation, and evaluation of the test procedure are the main topics discussed in Chapter 7.

The main test results are given and discussed in Chapter 8.

The final chapter, chapter 9, summarises the conclusions and the recommendations.

2. COAL OUTBURSTS - A REVIEW

2.1 COAL GAS RELATIONSHIPS

2.1.1 ORIGIN OF COAL ASSOCIATED GASES

The presence of the gases in the coal is generally related to the coalification process during which the original plant material has been subjected to pressure and moderate heat for long periods of time. Carbon dioxide and methane were produced during the early stages of the coalification process, as the coal material was buried, and the moisture content was reduced. As the process continued and the original plant material was transformed into coal, more methane and carbon dioxide was produced, together with smaller amounts of hydrogen, nitrogen, and some heavier hydrocarbons (such as ethane). Some of the gases so produced have been trapped in the coal under the confinement.

— There is no doubt about the source of methane; however, there are some doubts about the source(s) of carbon dioxide present in the coal. In addition to the carbon dioxide produced during coalification process, carbon dioxide may have been introduced to the coal by igneous activities. Some theories (Lidin, 1949) suggest that carbon dioxide was introduced to the coal after the coalification process had taken place. Lidin states that the gases which originated from coals during their metamorphism were composed of almost pure methane, having but insignificant admixtures of other

gases. Penetration of carbon dioxide into coal seams, according to Lidin, is connected with recent volcanic activity that injected large masses of basalts into the earth's surface during the Tertiary period. Gould and Smith(1980), studying C^{13}/C^{12} ratios of carbon dioxide in seam gases and in secondary calcites, also concluded that the source of carbon dioxide was external to the coal seam.

The occurrence of outbursts in the vicinity of intrusions where the main gas involved is carbon dioxide (Hargraves, 1983) appears to support these suggestions.

Whatever the source, methane and carbon dioxide are two important gas types that are commonly found in the coal seams. Usually the dominant gas is methane. However, in certain coal fields in the world notably in Australia, Czechoslovakia, and Poland, carbon dioxide concentrations are much higher than methane concentrations (Jackson 1984). In the Canadian coal fields methane is predominant (Patching 1970).

In a typical mining environment, methane exists in the free state in the fractures and pores of the coal mass. It also exists in the adjacent porous strata (about 5% of the total - Hargraves 1983). However, most of the methane is in an adsorbed state on the coal surfaces (as will be discussed later), with the result that the quantity contained in coal is much larger per unit volume than in adjacent strata.

2.1.2 POROSITY AND INTERNAL SURFACE AREA

Porosity is defined as the percentage of volume of the substance which is occupied by pores (voids). The pores which have a radius of less than $10A^\circ$ ($1A^\circ = 10^{-7}$ mm) are called micropores, the pores with a radius greater than $200A^\circ$ are called macropores, and those with a radius in between are called transientpores (Germanovich, 1984). Sometimes, for simplicity, the porosity of a substance is referred to, in the literature, in terms of micro and macro porosity avoiding the term transient porosity.

The porosity of coals can be as high as 40-50% in lignites (Berkowitz, 1985). It drops (Fig 1) to some 25-30% in subbituminous coals and to 1-3% in medium-low volatile bituminous coals, before increasing again among anthracites.

Coal of any rank (see Appendix 1.) possesses certain porosity. Generally, the porosity is the factor responsible for the capacity of coal to store methane or any other fluid. However, it is important to distinguish the pore volume from the pore surface area. The surface area, specifically the internal surface area, S , has a greater role to play in the capacity to adsorb gases, the swell in the presence of vapours and liquids, and development of heat on wetting (Van Krevelen, 1981). Generally the rate of reactivity in such processes is higher with higher internal surface area. The external surface area of coal, on the other hand, is significantly less than its internal surface area.

Thomas et al (1976) estimated the total surface area, S , for Illinois coals being as high as $292 \text{ m}^2/\text{g}$ while external surface area of the same coal was estimated as being only $1 \text{ m}^2/\text{g}$. Such high internal surface area is due to the fine structure of coal which may be considered as a compacted colloidal substance that has various sized interconnecting pores and capillaries.

The understanding that coal has a fine structure, and therefore has a large internal surface area, was not established until 1950s. As a result of work done during this period, several researchers (including Van Krevelen 1954, Anderson et al 1956) concluded that coal possesses a fine structure which contains two pore systems: A macropore system which is accessible to mercury under pressure, and a micropore system which, even at very high pressures, can not be permeated with mercury, but is accessible to helium at room temperatures. As a result of this development, it was possible to explain the continuing disagreement between measurements of internal surface area, as obtained with different methods (see Appendix 2). The CO_2 adsorption method is generally accepted and the results are less variable with temperature of measurement (Table 1).

TABLE 1

Surface areas of coal, as measured by different adsorbents and at different temperatures. (After Walker and Kini, 1965).

% carbon (d.a.f.)	Surface area ($\bar{S}$ m ² /g) measured				
	by N ₂ at -196	Kr -78	CO ₂ -78	CO ₂ 0	CO ₂ 25 °C
72.7	22	84	198	149	139
79.2	11	17	92	84	132
83.6	0	20	80	62	104
86.2	0	34	107	109	125
90.0	0	96	146	141	146
95.2	34	176	246	226	224

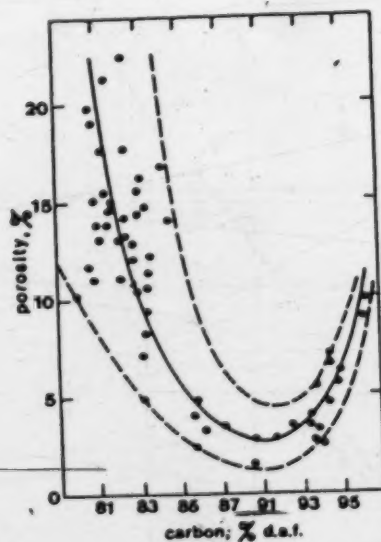


Figure 1: Variation of coal porosity with rank (After King and Wilkins, 1944)

2.1.3 FLOW OF GASES THROUGH COAL; GAS EMISSION

There are two basic mechanisms governing the flow of gases through coal (Cervic, 1969).

First, a flow through the fracture system which is defined by Darcy's Law:

$$Q = -(k \cdot A / \xi) \cdot (dP/dL)$$

Where ;

Q : The rate of flow , in cm³/sec.,

k : The fracture permeability, in Darcies,

A : Cross sectional area, in cm²,

ξ : Gas viscosity, in centipoise, and

dP/dL : The pressure gradient, in atm./cm.

The driving force for this type of flow is the pressure gradient.

The second flow mechanism is a molecular diffusion of adsorbed gas through the micropore structure. This is governed by Fick's Law:

$$Q' = - d \cdot A \cdot (dC/dL)$$

Where; d is the diffusion coefficient, in cm²/sec.

The driving force for for this type of flow is the concentration gradient (dC/dL) (in cm⁻¹).

The flow of the first type is more rapid than the second one (the flow through micropores is of a "bleeding" nature).

Although these two types of flow are separate and distinct phenomena, they are interdependent. Most of the gas escaping from a coal mass must diffuse to the fracture planes first obeying Fick's Law, and then it will flow along the fracture system obeying Darcy's Law. This interdependency has been described over a limited range of pressures by the following empirical equation (Anderson et al 1966):

$$C_0 = b \cdot P^\beta$$

Where; C_0 is the equilibrated concentration² of CH_4 in solid coal (cm^3 of CH_4/cm^3 of coal), and b , β are constants.

2.1.4 PERMEABILITY

Permeability is the ability of a material to allow fluids to travel within its body.

Under given conditions, the permeability is constant for a given material. Therefore, it is considered to be a parameter of the material. It can be determined by measuring flow rates through a specimen and by applying the Darcy's Law:

$$k = (\xi \cdot L \cdot Q) / (P_2 - P_1) \cdot A$$

where L is the length of the specimen, and $(P_2 - P_1)$ is the pressure gradient, and k , A , Q are as defined previously.

² In virgin coal, the pressure in the fracture system and the concentration of the gas in the micropore structure are in equilibrium. If the pressure is reduced, the equilibrium is broken and the gas begins to bleed from the matrix until a new equilibrium C_0 is reached that is the characteristic of the gas pressure in the fracture system.

However, Klinkenberg (1941), established that the gas permeability is also a linear function of reciprocal of mean pressure ($1/P_m = 1/(P_1 + P_2)$). Therefore he suggested that permeability found by applying Darcy's Law must be corrected by this factor (sometimes called Klinkenberg effect-Patching 1970), such that: $k_a = k \cdot (1 + (b/P_m))$, where k_a is called the apparent permeability and b is a constant. On the other hand, Patching (1970) states that while the effective permeability must be reduced because of the so called Klinkenberg effect, when the gas pressures are above atmospheric this effect is not large compared to the reduction due to the confining pressure. From his earlier work done on coals from Canmore Mines, Patching (1965) derived the following conclusions:

1. The effective permeability of a body of coal is due almost entirely to its fissuration.
2. Permeability is drastically reduced by the application of confining pressure.

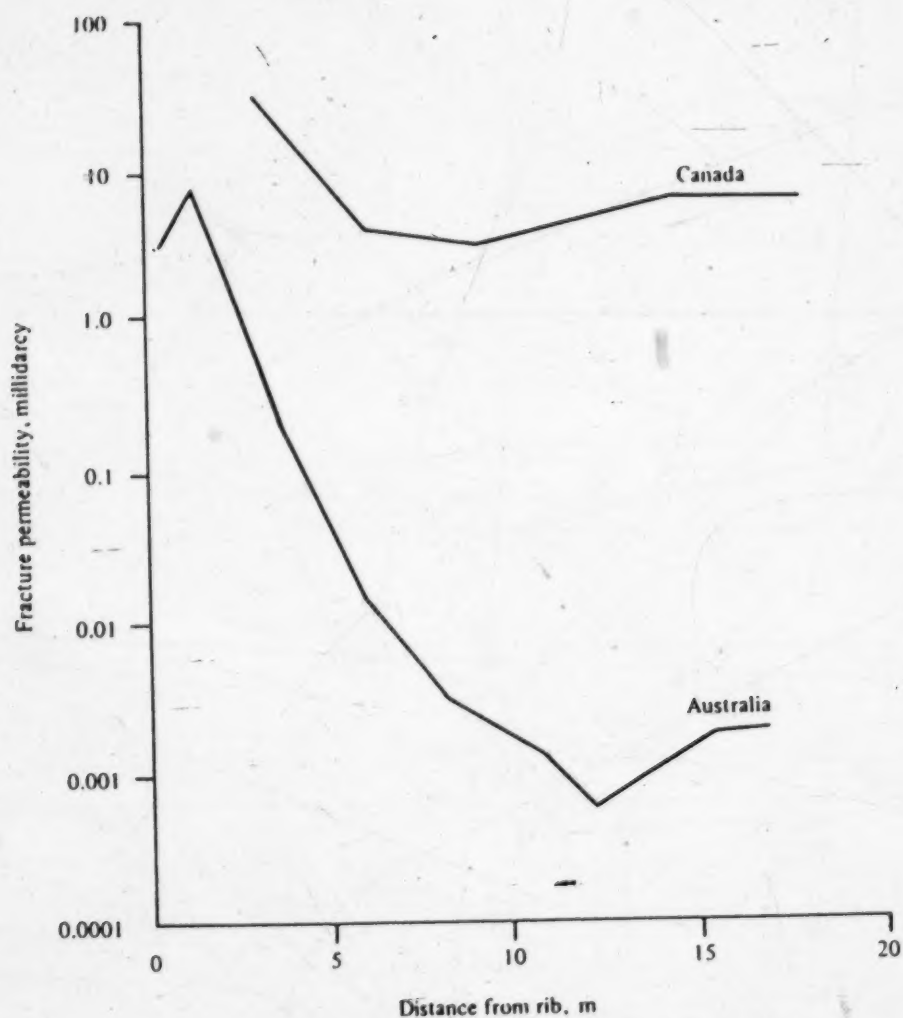


Figure 2 : Fracture permeability versus horizontal distance from the face. (After Fisekci and Barron ,1975)

Fisekci and Barron (1975) performed in situ permeability tests in Canmore Mines. They found that the permeability was minimum where peak abutment pressures might be expected (Figure 2). From this figure it is possible to draw similar

conclusions to those of Patching : The permeability is higher (About 20.0 md) where extensive fracturing is expected (close to face). It decreases rapidly, and reaches minimum (About 3.0 md) at about 9m from the face, where maximum abutment pressure is expected (refer to Canmore curve in Figure 2).

2.2 GAS SORPTION IN COAL

2.2.1 SORPTION

The term sorption has been used in the literature to refer the following range of physical-chemical processes.

- i) *Absorption* : A uniform penetration of one substance into the molecular structure of another substance (the process is reversible).
- ii) *Adsorption* : A surface effect, whereby one substance is physically held onto the surface of another (reversible).
- iii) *Persorption* : An extremely effective absorption of gas by a solid with the formation of an almost molecular mixture of two substances (irreversible), and
- iv) *Chemisorption* : An irreversible adsorption, whereby one substance is held to a surface of another by chemical forces.

Specific conditions; such as pressure, temperature, nature of substances involved etc. will determine the extent to which each of these processes can occur. When a coal and gas system is considered in the the context of the general

mining environment, chemisorption and persorption are ignored on the basis that they are irrelevant. Therefore, the sorption of gases on coal is considered as being totally physical in nature and reversible.

As noted earlier, the major portion of gas (such as methane) contained in coal is not held as free gas in the open spaces within coal, but as adsorbed layer along the surfaces of pores. Adsorption, thus becomes the more important of the reversible processes when dealing with the coal gas systems as encountered in mining.

2.2.2 ADSORPTION

The subject of adsorption is of primary importance in the field of surface chemistry. Therefore, many textbooks in this area give a comprehensive coverage of the subject. However, due to its importance in this study, and due to fact that not all the mining personnel are familiar with the subject, it is considered necessary to present an introductory review about the mechanisms and the basic theories of adsorption.

2.2.3 THE MECHANISM OF ADSORPTION

Adsorption occurs due to the unbalanced forces that exist on a surface. Generally there is a field of force around each atom and molecule of a substance which, by interaction, holds the molecules together, thus forming the substance. For example at the surface of a solid the average

number of atomic or molecular neighbours is only half as great as below the surface. Therefore, there is an imbalance of forces at the surface of the solid (called the adsorbent), and a marked attraction on the surface toward the atoms and molecules in its environment. This imbalance of forces can be neutralised by an attracted substance (called the adsorbate) such as molecules of an adsorbed gas.

The process of adsorption is spontaneous and continues until the free surface energy of the system, due to imbalance of forces, has reached a minimum. Van der Waals forces are considered to be responsible for adsorption. These forces arise from the interaction between the electrons of neighbouring molecules. Therefore, they are always present, but in ionic or homopolar lattices are overshadowed by electrostatic and valency forces respectively. They are much weaker than valency forces. Van der Waals forces are also associated with the condensation of liquids. Therefore, the energy change accompanying adsorption is of the same order of magnitude as that for the liquifaction of gases.

2.2.4 THEORIES OF ADSORPTION

There are many theoretical models which could be used to describe the adsorption of gases onto solids. The main difference between these models arises from the basic assumptions made, and each model can be accurately used under appropriate conditions. For example, the Langmuir

model (Langmuir, 1916) assumes that a monolayer of gas molecules occurs on the surface of solid during adsorption. On the otherhand, the Brunauer, Emmet, and Teller (known as BET) approach assumes the formation of additional layers (Brunauer, Emmet, and Teller 1938). While the formation of multilayers of gas is possible under high pressures, under the gas pressures encountered in the mining environment the Langmuir model is found to be the more applicable (the model is applicable for the pressures below 96.5 Mpa - Mikhail, 1971).

2.2.5 LANGMUIR THEORY AND EQUATION

This theory is based on the assumption that forces acting in adsorption are similar in kind to those involved in chemical combination. From the view point of the electron conception of the atom, the forces acting in ordinary chemical combination would be due to very strong deviations of the orbits of the outer electrons in the atomic structure, while the forces acting in adsorption would be due to comparatively small deviations. The theory assumes that elementary spaces or points of residual valency exist on the surface of the adsorbent. These spaces or points of residual valency can only be one molecule thick. In early stages of the adsorption process, only a fraction (θ) of the total surface has been covered by a monolayer of adsorbate. Thus the rate of evaporation from the surface is proportional to θ (i.e; $a_d \cdot \theta$). Similarly, the rate of

condensation of gas molecules onto the surface is proportional to the fraction of free sites remaining, or $(1-\theta)$, and the absolute pressure of gas, P , which determines the rate at which molecules contact the surface (i.e; $a_a \cdot P(1-\theta)$). At equilibrium these two rates are equal. Thus,

$a_d \cdot \theta = a_a \cdot P \cdot (1-\theta)$, and the fraction of the surface covered is:

$$\theta = a_a \cdot P / (a_d + a_a) = \lambda \cdot P / (1 + \lambda \cdot P) ,$$

where $\lambda = (a_a/a_d)$. Or,

$$1/\theta = 1 + 1/(\lambda \cdot P) .$$

Since the volume of the adsorbed gas (V) is proportional to θ , the above equation can be rearranged as:

$$1/V = 1 + 1/(\lambda \cdot P)$$

If $1/V$ is plotted against $1/P$, a straight line should be obtained which has a slope of $1/\lambda$, and intercept of 1. A good straight line indicates that the results conform to the Langmuir isotherm.

2.2.6 HEAT OF ADSORPTION

The amount of adsorption decreases as the temperature increases (Fig. 3 - Page 24). This implies that the process

is an exothermic one. This can be seen by the reference to well known thermodynamic equation:

$$\Delta G = \Delta H - T \cdot \Delta E_n$$

relating the changes ΔG , ΔH , and ΔE_n in the free energy, the heat content, and the entropy respectively which occur during a physico-chemical process such as adsorption. If the adsorption occurs spontaneously, then the free energy must diminish during the process so that total free energy, ΔG , must have a negative value. Moreover, ΔE_n will be negative because the molecules of adsorbate must have less freedom in the adsorbed state than in gaseous state. Consequently by the above equation ΔH will have to be negative, i.e the process is exothermic.

Desorption, on the other hand, is an endothermic process.

2.2.7 THE ADSORPTION OF CO₂ AND CH₄ ON COAL

The amount of gas adsorbed x , by a given adsorbent depends on the pressure, P , the temperature, T , the nature of gas, and the nature of solid. Thus:

$$x = f(P, T, \text{gas}, \text{solid})$$

where the form of the function f can be highly complex.

It is possible, however, to examine the effect of the parameters involved by varying each in turn whilst keeping

the others constant. Thus, for a given gas and solid, two useful graphical representations for adsorption capacity x , can be obtained. Namely,

- 1) $x = f(P)$, $T = \text{constant}$, (adsorption isotherm),
- 2) $x = f(T)$, $P = \text{constant}$, (adsorption isobar) .

A large amount of data, in the form of isotherms and isobars, has been obtained over the years so that reliable empirical formulation of these graphs is possible. One such empirical relationship, called Gibbs adsorption equation, will be used later (Section 4.3.1) to relate changes in surface energy due to adsorption.

The effect of rise in temperature is always to reduce the adsorption at a given pressure for a given adsorbate-adsorbent pair as shown in Fig. 3 for the adsorption of methane on coal (Kim, 1977). The effect of rise in pressure, on the other hand, is to increase the adsorption (Fig. 4 - Kim, 1977). From these two figures it can be seen that the temperature adsorption relationship is linear, while the rate of increase in adsorption tends to decrease after approximately 5 Atm. as the pressure increases at constant temperature.

The adsorption capacity of a given gas for a particular adsorbent, at a given pressure and temperature, depends also on the molecular size, the liquefaction temperature, and the polarity of gas molecules (Gregg 1961, Ettinger 1966). The

molecular size is important from the perspective that "all adsorbents depend upon internal surface area for the full use of their adsorption capabilities. If the molecular size is too large, adsorption will be hindered and adsorption capacity will decrease as very large molecules block or can not penetrate pores or pathways within the adsorbent."

(Bernardin, 1985).

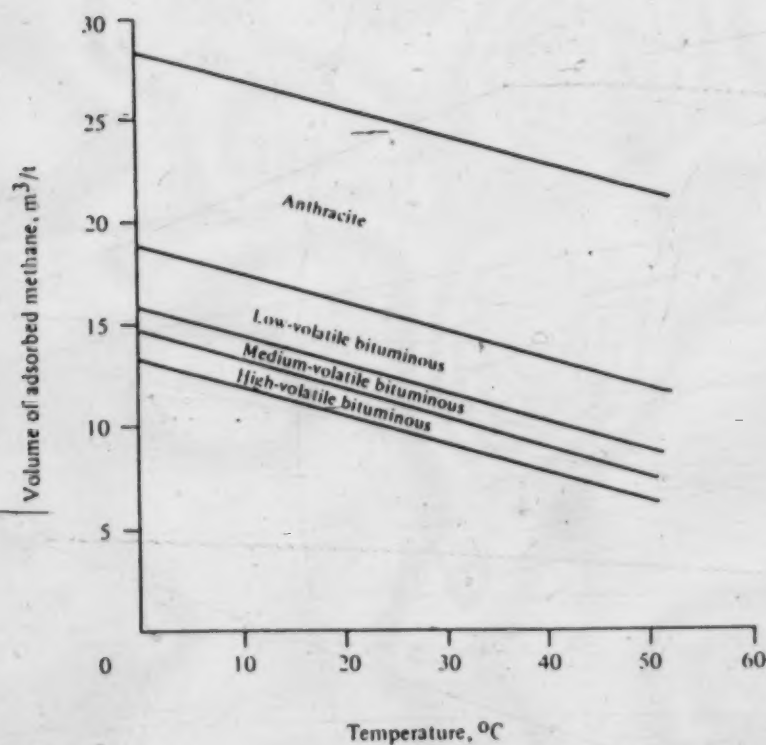


Figure 3 : Variation of coal adsorption capacity with temperature at 10 Atm. (After Kim, 1977).

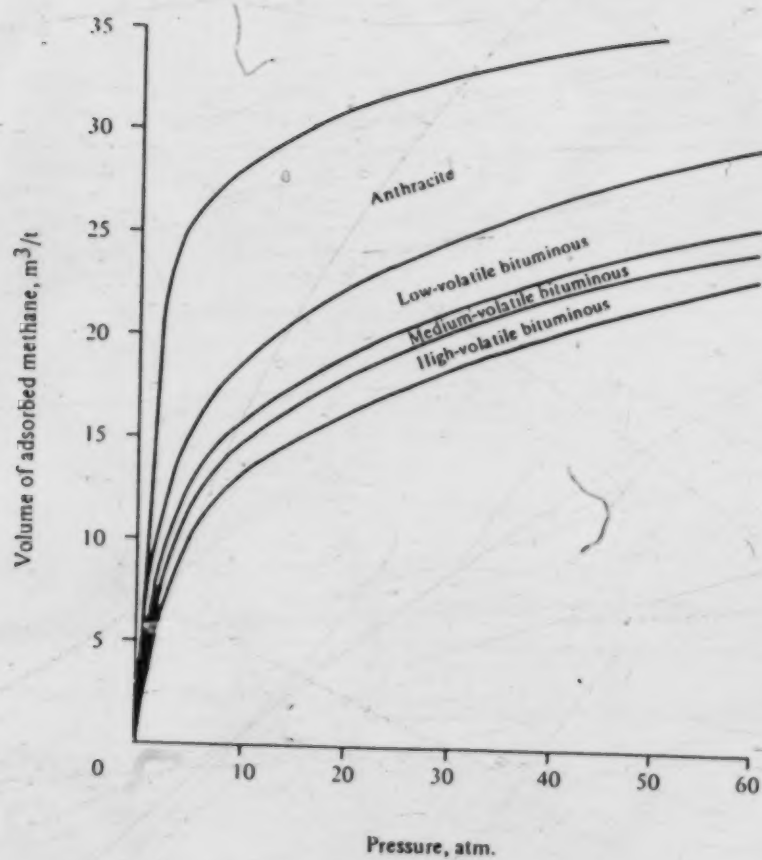


Figure 4 : Variation of methane adsorption isotherm (0°C) with pressure. (After Kim, 1977).

The polarity of gas molecules is also of significance. The potential for an ionised molecule of gas to be adsorbed is greater than that of a neutral one (depending on the nature of solid).

The effect of polarity of the gas molecule on adsorption becomes more important in the oxidation zone. This zone may extend from surface down to about 1000m into crust, depending on the tectonic history of area. In this portion of crust weathering takes place due to actions of some agents such as water, oxygen, carbon dioxide penetrating down from the earth's surface.

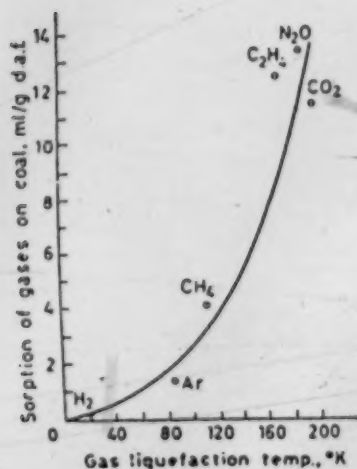


Figure 5 : The relation between sorption of gases on coal (at 30°C and 1 atm.) and gas liquefaction temperature (After Ettinger et al, 1966).

There is a parallelism between the amount of adsorption and the liquefaction temperature. Figure 5 shows the amount of gas adsorbed on coal for different types of gases. From this figure it can be seen that CO_2 , which has a higher liquefaction temperature than methane, has almost twice the adsorption capacity of methane. This is attributed to the fact that at this temperature methane molecules do not possess enough activation energy to penetrate the fine pore structure of the coal. (Ettinger et al, 1966).

The physical properties of the solid such as surface area, size and distribution of pores, directly affect adsorption by determining the amount of adsorbent capacity available and molecular size which can be adsorbed.

2.3 MECHANISM OF COAL OUTBURSTS

2.3.1 MECHANISM OF COAL OUTBURSTS

The fundamental physical or chemical process involved in the occurrence of outbursts has been a major subject for investigators for almost a century. Several theories have been developed in efforts to explain the outburst phenomenon. The successful theory, however, must explain the events leading up to and occurring during an outburst. The events that may precede an outburst are summarized by Shepherd et al (1981) as:

1. The occurrence of distinct, audible noises originating close to the mine opening.

2. The instability of the coal face, which is usually manifest as a bulge or outward push of face in the opening.
3. A temperature decrease in the solid or in the mine atmosphere. Coeuillet (1964) mentioned Polish measurements in coal saturated with CO₂ in which a drop in temperature of several degrees in the solid was a precursor of an outburst³

The theory must also be able to explain the projection of fine coal to a considerable distance (up to 100m., as noted earlier). The severity of carbon dioxide related outbursts as compared to those involving methane is another point to be considered in the theory.

There have been two basic theories put forward for explanation of the outburst phenomenon:

The oldest theory is known as the "Pocket Theory" or the "Nest Theory" (Ruff, 1930). According to this theory there exist, within the coal seam, localised pockets of low strength, crushed, coal that are saturated with gas. If one of these pockets is punctured during the advance of the face, a large volume of adhesionless coal particles floating in gas is displaced with the dynamic effect of the high gas pressure.

This theory has been opposed (Jarlier, 1936) on the basis that "no changes could be established in the

³
A temperature decrease is not often noticed and the instability of the coal face may be just in the form of loosening of small portion of coal on the face.

composition of materials at places where outbursts have occurred or where these have been absent" (Lama, 1980). Where the frequency of outbursts is higher, one would expect that such pockets should be more frequent. According to Lama there has been no substantial evidence to confirm this expectation.

The second and more commonly accepted theory is known as the "Dynamic Theory" (Khodot, 1961 and 1964). According to this theory, rock pressure causes fracturing of coal and the release of large volumes of gas. The gas is instrumental in the displacement of coal. Therefore both the gas and the rock pressure play an equal role in the mechanism of an outburst.

Among the dynamic theories put forward, the theory of Khodot is worthy of attention. It is possible to explain the several phenomena that precede or occur during the coal outbursts. The theory will be explained in greater detail in the following section.

According to Khodot (1964) an outburst occurs in three stages :

Stage one (Trigger Stage)

Under conditions of three dimensional compression, coal has some elastic properties and is capable of accumulating large reserves of potential energy. At the same time its resistance to rupture, especially in cases of complex loads i.e at face front, is not high. In cases of the sudden opening of some areas subject to

great pressure of rock and gas, coal elastic forces (sometimes also the dynamic load of the roof in secondary workings) crush the coal at face, and shift it towards the excavated space, thus resulting in an increase in coal fissuring and the release of gas from the system of initial pores and fissures. Accordingly, the force that caused displacement of coal towards the face is further enhanced by the high gas pressure gradient effective on opened fissures, and by increase in the ratio of open fissures to total area of fissures.

Stage Two (Free Gas Stage):

The coal displacement, which began under the effect of rock pressure, develops into ejection due to the participation of free gas. (i.e the gas which is not sorbed onto coal).

Stage Three (Adsorbed Gas Stage):

As the coal is disintegrating, favorable conditions are created for intensive desorption of the gases from the fine pores. The process is accompanied by a very intensive gas release. This stage of an outburst is characterised by formation of a flow of finely crushed coal suspended in gas. The greater the initial coal disturbance, the more intensive the degree of crushing, and higher the initial gas content then the stronger will be the flow of finely crushed coal.

To summarise the effect of gas in the occurrence of coal outbursts, Khodot draws an analogy with "fuse-charge" effect in an explosion such that: No matter how great is the potential energy of the charge (gas action), it can not be released without a fuse (stress in the case of an outburst).

Comparing the two theories, it can be assumed that once the coal is crushed under the effect of stress, i.e. after the first stage in Khodot's theory, the conditions created can be regarded as similar to the pocket theory conditions.

The temperature decrease in the solid, which is reported to occur prior to outburst, but not confirmed in general, can be attributed to two factors:

1. Gas expansion.
 2. The endothermic nature of the desorption process.
- While free gas is the responsible factor in the first case, adsorbed gas is the factor involved in the second case.

The adsorption of gases on coal also has its contribution on the outward push of the coal face which often shows itself just loosening of small coal on the face. (i.e. due to swelling caused by adsorption - Gregg, 1961)

The occurrence of audible noises originating close to the opening may be attributed to the crushing of coal (crack initiation and propagation). Because the majority

of outbursts occur in the vicinity of fault zones (Marshall et al 1980, Hargraves 1983, Jackson, 1984), this may also be due to a minor slip in the fault plane under the abutment pressure.

The reason for the higher intensity of carbon dioxide associated outbursts is twofold:

1. The adsorption capacity of coal for carbon dioxide is 2-3 times greater than methane.
2. The desorption rate of carbon dioxide is higher compared to desorption rate of methane from coal (Singh, 1968).

There are different opinions about the relative influence of the role of stress and of gas in coal outburst mechanisms. Gray (1980) states that the failure is due to the stress and strength characteristics of coal, fluid pressure being an operative stress. The failure may be brought about totally by a stress combination exceeding the strength of coal, at one extreme, without the influence of gas, or at the other extreme, the internal gas pressure is sufficient to exceed the tensile strength of coal in an unconfined state. Any combination in between the extremes may exist. Hargraves (1983), on the other hand, comments that, by definition, outbursts must involve the release of gas and that failure without the participation of gases is an "rock burst". There is no doubt that the failure is due to stress-strength characteristics of

coal.

Gray (1980) uses the Coulomb-Navier failure theory:

$$\tau_f = \rho + \sigma \cdot \tan \phi$$

Where:

τ_f = The shear stress at failure,

ρ = Cohesion of the material,

σ = The effective stress normal to the plane of failure,
and

ϕ = The angle of internal friction.

When there is pore pressure P , within the coal mass, then the equation becomes:

$$\tau_f = \rho + (\sigma - P) \tan \phi,$$

i.e the pore pressure reduces the strength of coal, and consequently it will fail at a lower applied stress value than without pore pressure.

Another aspect of coal outburst phenomenon is the projection of fine coal out from the face. This requires a considerable amount of energy. Farmer and Pooley (1967) estimated that 10^9 Joules of energy is released during an outburst involving 50 tonnes of coal of gasiness 30 m³/t. Khodot (1964) lists the sources of energy involved as:

1. Potential energy of rock deformation,

2. Kinetic energy of rock movement,
3. Potential energy of coal seam deformation,
4. Energy of coal seam position (steep, or medium dip), and
5. Intrinsic energy of gas.

Among these, the items 1,3,4 and 5 are likely to be the most important sources of energy involved in an outburst.

Based on the above factors, Khodot found that the potential energy of coal deformation released during the face advance is sufficient only for the displacement of part of a coal seam towards the excavated area and for its breakdown along the largest fissures mainly of tectonic origin (work required for this process was found to be 0.065 MJ/tonne (6.6 t.m/t). However, the expansion energy of methane released during coal fracturing may reach to 5.9 MJ/t (600 t.m/t) of coal. This amount is sufficient both for coal ejection and for coal crushing to small fractions including the so called "bull dust".

Gray (1980) considers the sources of energy as:

1. Gas strain energy, and
2. Rock/coal strain energy.

In both cases the strain energy is converted to kinetic form by change of strain during an outburst. Figure 6 illustrates the rock strain energy in a one dimensional rock burst situation. In this figure, two

- possible unloading curves are shown : a) A relatively slow unloading - where the unloading curve may reach equilibrium with the unloading curve of the "normal" material curve with the result of release of certain amount of energy. b) A faster unloading - Where a non equilibrium condition is created, thus allowing much higher energy release, resulting in a possible rock bursting situation.

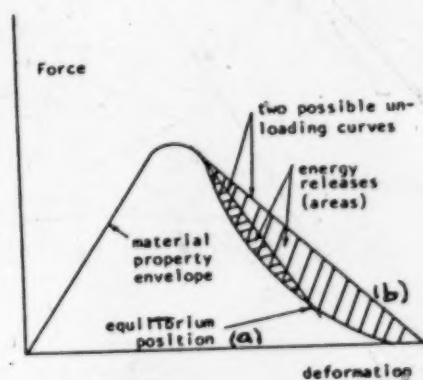


Figure 6: One dimensional rock bursting. (After Gray, 1980)

Gray estimated the gas strain energy and rock coal strain energy released during an outburst which occurred in Leichard Colliery, Australia to be 219 MJ. and 30 MJ. respectively (400 tonnes of coal, 300 m³ gas). These figures show the relative importance of gas energy in the displacement of coal during the outburst.

2.3.2 THE FACTORS AFFECTING THE OCCURRENCE OF THE OUTBURSTS

Apart from the factors discussed so far such as permeability, porosity, and internal surface area of the coal, the content and type of gas; other factors influencing the occurrence of outbursts are the depth of cover, the geological structure and the gas pressure.

DEPTH OF COVER:

The occurrence of outbursts are reported at depths of as little as 80m. (Belin, 1964). In Australia, it has been found that outbursts can occur at depths as shallow as a 100m., when a geological disturbance exists (Lama, 1980). However, in the absence of a shear zone, Lama states that generally a depth approaching 500m is required. In Canadian coal fields the outbursts occurred below 200m (Patching and Botham, 1966). These examples show that there is no common minimum depth for the outbursts to occur.

As the depth of mining increases, the vertical component of the stress increases according to relation : $\sigma_v = \eta \cdot h$ (where σ_v is the vertical stress, η is the unit weight of rock, and h is the depth). The stress, as noted earlier, has a critical role in the mechanism of an outburst. Therefore, it is reasonable to assume that the outburst problems should get worse as the depth of mining increases. In Canmore area, for example, the frequency of outbursts was found to increase with depth (Patching, 1986).

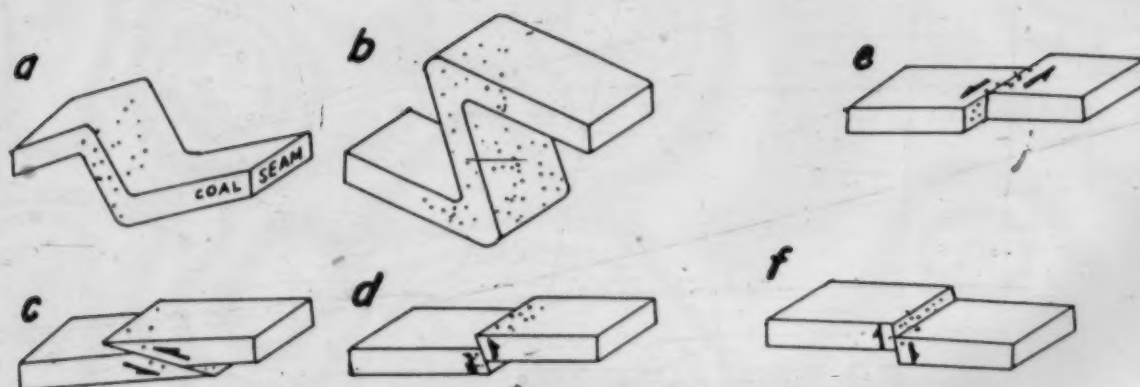
On the other hand, the total stress field is not defined by the vertical stress alone, and it is differential stress, not the total stress that is responsible for the failure of the structure.

Although there are indications that the incidents of outbursts increase with the increase in the depth of workings, it appears that no general rule for the relationship with depth is universally applicable (Jackson, 1984). Depth of cover, however, is a contributing factor to the stress field and therefore to the occurrence of outbursts.

GEOLOGICAL STRUCTURE:

Almost all the reports dealing with the subject, emphasize the importance of the relationship between outbursts and geologically disturbed areas. Usually, there is a regional or local fault or fold present in the vicinity of an outburst site (See Figure 7). The strike slip fault regions are reported to be more susceptible to outbursts as compared to other structural features shown in this figure (Shephard et al, 1980)

Faults are sometimes accompanied by intrusions of porphyry (in Poland), or clay and sandstone (U.K). The latter have formed resistant cores against which the seam has been crushed in subsequent shifts (Belin, 1964).



a: Asymmetrical anticline, b: Recumbent fold, c: Thrust fault
d: Reverse fault, e: Strike slip fault, f: Normal fault.

Shaded zones represent outburst prone coal.

Figure 7 : Schematic block diagrams of geological structures associated with outburst sites. (After Shepherd et al, 1981).

It has been reported that (Belin, 1964) in Balkan coal fields of Bulgaria, 90% of outbursts occurred in tectonically disturbed zones. In the South Wales area of U.K., geological disturbances have been associated with all recorded incidents of outbursts, although the presence or full extent of disturbance may not have been revealed until after the outburst (Swift, 1964). Swift also emphasizes the negative effect of intrusions which bring about unequal

stress distributions ,and thereby increase the liability to outbursts. The dangerous points are considered to be at the approach and departure from the neck of intrusions.

In Australia, the great majority of outbursts that have occurred were also associated with the fault zones; either major faults or shear zones (Hargraves, 1983; Marshal, 1980; Lama, 1980). In Canada, the Crowsnest Pass and Cascade coal areas are also known as geologically disturbed areas (faulting and folding).

Lama (1980), emphasized the effect of geological structure by giving global scale examples. This author concluded that outbursts are probably associated with regions of high geological activity on a regional level; such as compressional forces causing mountain building and regional up-lift (Czechoslovakia, Yugoslavia, Turkey, India, Japan), large erosion and marked changes in the topography (Canada, USA, Poland) ; or high localized compression as in salt mines in Germany and Poland. These examples indicate the possibility of high lateral stresses associated with the occurrences of outbursts.

It appears that lateral stresses may be of a higher magnitude than the vertical stresses in the vicinity of fault zones. For example, Williams and Rogis (1980) quote lateral stresses of 10 MPa. with vertical stresses of only 2.2 MPa. at depth of 240 m. in parts of Collinsville States Mines in Queensland, Australia. The geometry of the geological structure is important when considering lateral

stresses. Fault planes will transmit the normal stress component virtually unchanged, whereas, the tangential component will be transmitted according to the degree of friction along the fault plane (Shepherd et al, 1980). This conclusion has been established experimentally by the studies done in The Colorado School of Mines experimental mine. These studies (Abel and Lee 1973, Lee et al 1969) found that the stress orientations were controlled by the fault plane. The same workers also found that the magnitude of average horizontal and vertical components of the in-situ stress field was more than twice as large as predicted from the overburden.

From these considerations it is possible to conclude that geological disturbances, specifically fault zones :

- a) Are regions of high stress fields,
- b) Control the direction of major principal stress by their geometry. At shallow depths the major principal stress is in the horizontal direction (which may be towards the face of the underground opening),
- c) Weakens the coal, so that it fractures more easily.

These factors will help to initiate failure in the coal mass. The fault zones may also act as gas conduits. High stress fields coupled with gas concentration and pressure, as explained earlier, cause an outburst. It appears that geological structure of coal deposits has more significance in the occurrence of outbursts than the depth of workings.

GAS PRESSURE:

The effect of pore pressure on the strength of rocks can be explained in terms of the law of effective stress which was introduced by Terzaghi (1945). According to the law of effective stress, the deformation and failure of a material are controlled by effective stress which is defined as:

$$\sigma' = \sigma - P$$

Where σ' is the effective stress, σ is the applied external stress, and P is the pore pressure.

In terms of principal stresses the above equation becomes:

$$\sigma'_1 = \sigma_1 - P$$

$$\sigma'_2 = \sigma_2 - P$$

$$\sigma'_3 = \sigma_3 - P$$

and, Colulomb - Navier failure criterion takes the form:

$$\tau_f = p + \sigma' \tan \phi$$

Terzaghi's effective stress concept is well established. Several researchers (includin Handin et al, 1963) showed that the criterion for failure in terms of effective stress holds reasonably well provided that the permeability is sufficient to allow the movement of the fluid and that a connected system of pores exist and also

that the fluid is inert, so that the effect is purely mechanical.

The effect of pore pressure can be illustrated by the following figure (Jaeger and Cook, 1969):

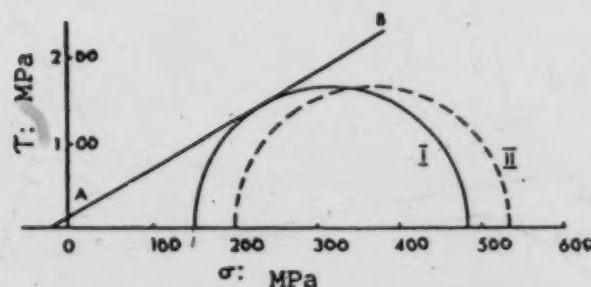
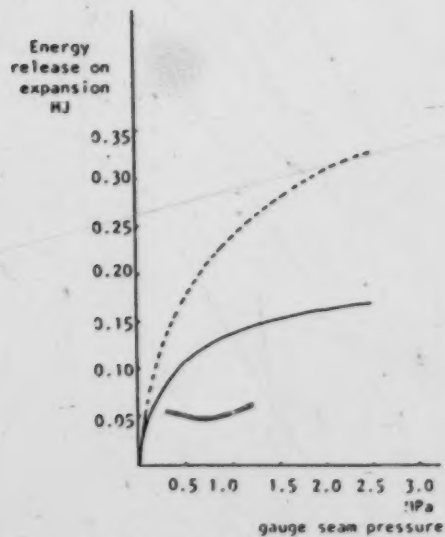


Figure 8 : The effect of pore pressure on failure.

Here AB is the experimentally determined Mohr envelope for zero pore pressure. Curve I for $P = 50$ Mpa corresponds to failure with $\sigma_1 = 540$, $\sigma_2 = 200$ Mpa so that $\sigma'_1 = 490$, $\sigma'_2 = 150$ Mpa, and Mohr circle is seen to touch AB. Curve II which shows the Mohr circle for $\sigma_1 = 540$, $\sigma_2 = 200$ Mpa with zero pore pressure, is seen lie inside the Mohr envelope. As the pore pressure is increased this curve is moved to the left until it touches the Mohr envelope, and failure can take place.

The gas pressure has a critical role in the initial failure of the coal mass, in the flow characteristics, and in severity of outbursts:

1. Failure of coal mass: As discussed earlier, the higher the gas pressure, the lower is the resistance of coal mass for failure at a constant stress level.
2. Flow characteristics: The quantity of gas depends on the pressure gradient (Darcy's Law), and the gas concentration (Fick's Law).
3. Severity of an outburst: The amount of energy released is directly proportional to the gas pressure as shown in Figure (9).



— Adiabatic
- - Isothermal

Figure 9 : Energy release on expansion vs. pressure. (After Gray, 1980).

MOISTURE CONTENT:

The effect of high moisture content of coal is to decrease its capacity to adsorb gases. The relationship is given (Ettinger, 1957) by the following empirical formula:

$$\frac{V_h}{V_d} = \frac{1}{1 + 0.3w}$$

Where,

V_h is the total sorption capacity of humid coal,

V_d is the total sorption capacity of dry coal, and

w is the moisture content in percentage.

COAL RANK:

In general, high rank coals such as anthracites and low volatile bituminous coals are more susceptible to outbursting than coals of lower rank (Jackson, 1984). The main reason for this is that the capacity of coal to adsorb gas increases with increasing rank (as defined by volatile matter percent). Hargraves (1983) suggests a threshold value of 30% volatile matter (d.a.f) for Australian coals. Coals with a volatile matter content less than this are regarded as "outburst-prone".

LOW PERMEABILITY OF COAL/STRATA:

The low permeability of coal and/or strata surrounding the coal seam results high pressure build-ups behind the face, thus contributing to the state of stress.

STRENGTH OF COAL:

The gas pressure, as discussed earlier, contributes to the state of stress. However, there are some indications, on theoretical basis and on the basis of some indirect experimental studies, that the presence of gas in the coal seam under pressure also affects the strength of coal. It is believed that the effect (if any) of gas in coal (other than the mechanical effect) is due to adsorption of gas molecules on coal surfaces. Presence or otherwise of such an effect is the main subject of investigation of this thesis.

3. STATEMENT OF THE PROBLEM TO BE INVESTIGATED/OBJECTIVES

One of the basic assumptions involved in the law of effective stress is that the fluid must be inert, i.e., it must not affect the surface properties of solid. However, the gases involved in outbursts, methane and carbon dioxide, are not inert; they adsorb on to the coal surfaces (as discussed earlier). This adsorption process should theoretically change the surface properties of coal and thus should influence the coal strength.

It is therefore important to determine whether or not the adsorption of mine gases on to pore surfaces of coal produces a deviation in the law of effective stress, and whether such a deviation is significant.

The aim of this study is to demonstrate, experimentally, the validity or otherwise of the law of effective stress in problems where the fluid involved is not inert; specifically the effect of adsorption of mine gases on the strength of coal. This will be verified by measuring the strength of coal specimens in the laboratory using inert gases (such as helium) and adsorbing gases (such as carbon dioxide) over a range of pore pressures. These tests should indicate whether or not the postulated deviation from the law of effective stress is significant.

4. ADSORPTION AND STRENGTH OF MATERIALS

4.1 PREVIOUS INVESTIGATIONS

There has been considerable work done on the strength reduction as related to the adsorption of fluids in general. The materials tested are usually metals, silica rods, and glass slides. The adsorbates used include water vapour, ethyl alcohol, benzene vapours etc. Reference to such works can be found in Sereda et al, 1967, Gregg, 1961 and others. A short list of such works is given in Table 3 (Page 59). The work of Rehbinder and Lichtman (1957) is the one usually quoted, and the effect is sometimes referred to as "Rehbinder effect" (Sereda et al, 1967). These authors tested silicate glasses (and several metallic materials) under constant load with different solutions (such as water, alcohols etc.). Their results showed that resistance of solids to strain and rupture was reduced by adsorption from the surrounding medium, and that the phenomena was not connected with dissolution or corrosion, but was associated with reduction of free interfacial surface energy by reversible adsorption. According to Rehbinder and Leichtman, the weakening effect due to adsorption will take place on the simultaneous fulfilment of two conditions:

1. The state of stress should aid the development of weak spots (ultramicrocracks) in the surface layers of the solid; i.e, high tensile stresses (super stresses) should be present.

2. During the time of development of the defects the surface energy over a sufficiently large portion of newly formed solid surface should be lowered by penetration of surface active components from the medium in the form of adsorption layers.

The time factor also seems to be important in these experiments, because the greatest adsorption effect was found by the prolonged action of stress on the solid surface.

Considering the low tensile strength of coal and high abutment pressures in front of a coal face, it is not difficult to envisage the development of weak spots on the surface layers of coal. The lowering of surface energy, as will be demonstrated by an example, may be significant in coal. There are also indications that the lowering of strength is somewhat time dependent. Patching (1986) comments that a face can stand for days without a sign of an outburst, but as soon as work resumes the outbursts can occur. Hargraves (1980) noted the record occurrence of outbursts on Mondays.

Ettinger and Lamba (1957) studied the effects of the gas medium in the coal breaking process. The technique employed was based on the "pounding method", in which a weight is dropped from a constant height on coal. The amount of dust produced of a size less than 0.5 mm. was taken as the hardness index. "The hardness index was determined in air and in carbon dioxide at a pressure of 40 atmospheres."

They found that coals with a few cracks possess equal hardness in air and in carbon dioxide. However, the hardness of disturbed coals in carbon dioxide may be half that in air; the mean reduction being by a factor of 0.75. These authors reported a similar but smaller effect with methane. Their interpretation of these results was that "when coals experience mechanical action in a gas medium, destruction effects not only large surfaces, but also micropores which cannot be destruction surfaces when no gas is present. On account of the reduction in free surface energy at adsorption, the gas penetrates in to the tectonic and endogenetic 'embryo' micropores and causes additional deformation by a wedging action of adsorption layers. However, where there is no such initial destruction in coal, the gas itself cannot cause the appearance of new coal/gas dividing surfaces and thereby facilitate the destruction of coal along new planes."

Khodot (1964) makes a similar statement, stating that even in very gassy seams outbursts cannot occur without the triggering mechanism of stress. If localised gas pressure zones exist, however, the sudden relief of the pressure can cause outbursts as demonstrated by Botham (1957) in his laboratory experiments.

Tankard (1957) obtained the change in surface area of coal in a CO₂ gas environment as compared to the surface area obtained in air, under the same impact energy (drop weight method). Comparisons were made between the surface

area generated in air (1 atm.), and in CO₂ (10, 20, 30, 40, and 50 atm. levels). On the assumption that change in surface area generated is an inverse measure of the strength of coal, it was concluded that when compared with the results obtained in air (1 atm.) there was a strengthening of coal around 10 atm., and that coal at 40 atm. was considerably weaker than when tested at 20 atm. These differences were attributed to the effect of adsorption on the strength of coal.

Due both to the experimental method employed, and to the interpretation of the results, the above conclusion is questionable (see Section 4.2).

Tankard's work is directly related to the present study. Therefore, a more detailed review and analysis of the work is presented in a separate section (Section 4.2).

Desai and Gregg (1958) tested saturated coal compacts for modulus of rupture using water vapour, benzene, acetone and methanol as saturation fluids. They found that in some cases the modulus of rupture fell to less than a third of the value it had in vacuum.

Boozer, Hiller and Serdengecti (1962) performed extensive triaxial tests with different interstitial fluids. They found that the deformation behaviour of limestone and sandstone were affected by fluids which are strongly adsorbed on grains of these rocks. The predominant effect of these fluids was a decrease in the ultimate strength (sandstone), and yield stress (limestone). For example, a

16% decrease was observed in the ultimate strength of sandstone when tested with Oleylamine as compared to n-hexadecane. The authors concluded that strong adsorption of these fluids on the surfaces decreases free surface energy of individual crystals of these rocks, and thereby reducing the strength.

Colback and Wiid (1965) studied the influence of moisture content on the compressive strength of rocks. They found that the uniaxial compressive strength of quartzitic shale specimens was considerably affected by the moisture content of the sample (Fig. 10). The authors concluded that the reduction in strength with increase in the moisture content was primarily due to reduction in uniaxial tensile strength which in turn is a function of the molecular cohesive strength of material:

$$\sigma_m = (2\gamma E/a)^{0.5}$$

where σ_m is the molecular cohesive strength of material, γ is the surface free energy of material, E is the Young's Modulus, and a is the spacing between neighbouring atomic planes.

Falkenberg (1972), studying the role of additives in comminution, showed experimentally that, with 44 micron particle size, the grinding efficiency is increased when oxygen is used instead of air (22%), and further increased (27%) when an organic solution (called TDA solution) is

used. The increase was less significant with larger particle sizes. The author concluded that the improvement was due to adsorptive effect of grinding additives which help to propagate microcracks.

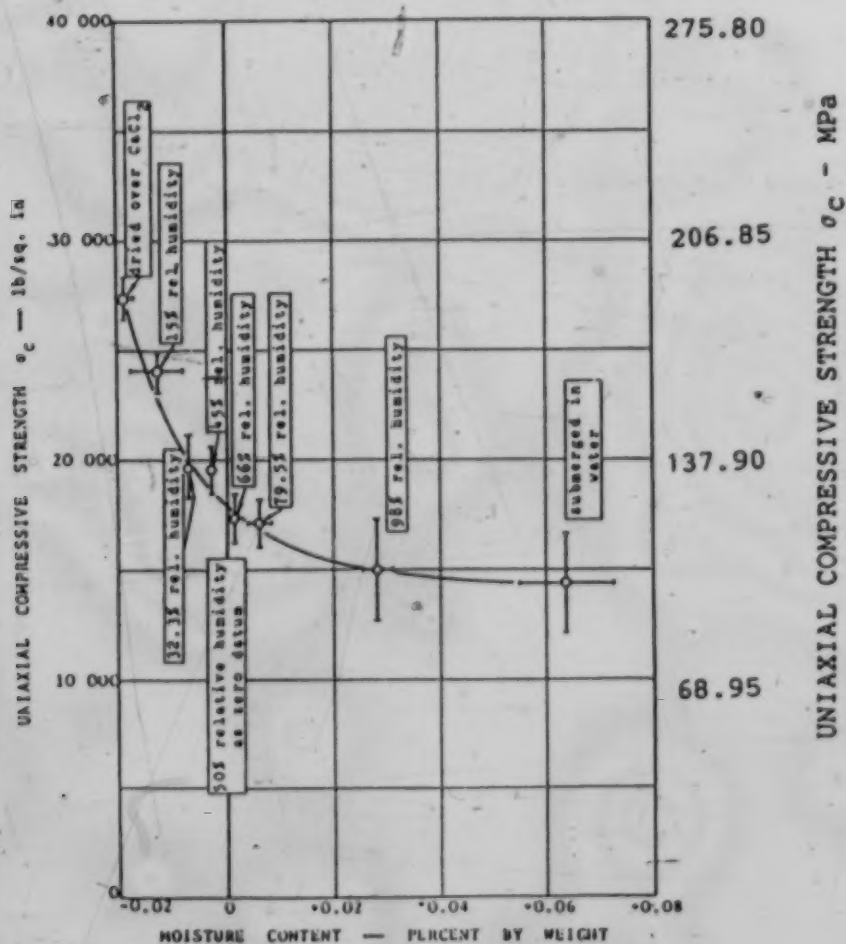


Figure 10: Relationship between uniaxial compressive strength and moisture content for quartzitic shale specimen (After Colback and Wiid, 1965).

4.2 REVIEW AND ANALYSIS OF THE WORK DONE BY TANKARD, J. H. G. (1957)

The most thorough investigation of the effect of sorption on the strength of coal, is the work done by Tankard (1957). A brief review and analysis of this work is presented below.

Procedure followed: A 10 g coal sample of -10 mesh size was placed in the "Sorption Bomb". The carbon dioxide gas was allowed in at the appropriate pressure and the sample was left in for a sorption period of 20 min. The sample was then taken out and transferred to a drop-weight crusher where a constant impact energy was applied via a drop weight. The crushed sample subsequently was transferred to an air permeability apparatus and the change in the surface area ($\Delta S = \Delta S_{\text{gas}} - \Delta S_{\text{air}}$) was calculated from the air permeability test results.

This change in surface area was taken as an inverse measure of the strength of coal, such that a higher change in the surface area indicated a lower strength for the coal tested. Four different Australian coals were tested in this manner at five pressure levels (1, 10, 20, 30, and 40 Atmospheres). Each test was repeated six times.

Results obtained: The results of this work has been summarised in Table 2 (with the standard deviations being calculated from Tankard's data and added by author of the present study), and are plotted in Figure 11.

TABLE 2

The change in surface areas of coal (in cm^2) as obtained after dropping a constant weight on coal (Tankard, 1957). Tabulated as mean and the standard deviation of the mean (with st. dv. given in parantheses).

PRESSURE (Atm.)	COAL TYPE			
	Collinsville	Excelcior	Metropolitan	South Bulli
1 (CO_2)	308 (36)	319 (55)	220 (51)	244 (42)
10 (CO_2)	269 (35)	296 (31)	223 (31)	223 (48)
20 (CO_2)	334 (39)	299 (31)	208 (34)	242 (66)
30 (CO_2)	302 (39)	377 (93)	280 (20)	290 (35)
40 (CO_2)	371 (41)	384 (66)	292 (55)	266 (27)
1 (Air)	317 (?)	311 (?)	220 (?)	240 (?)

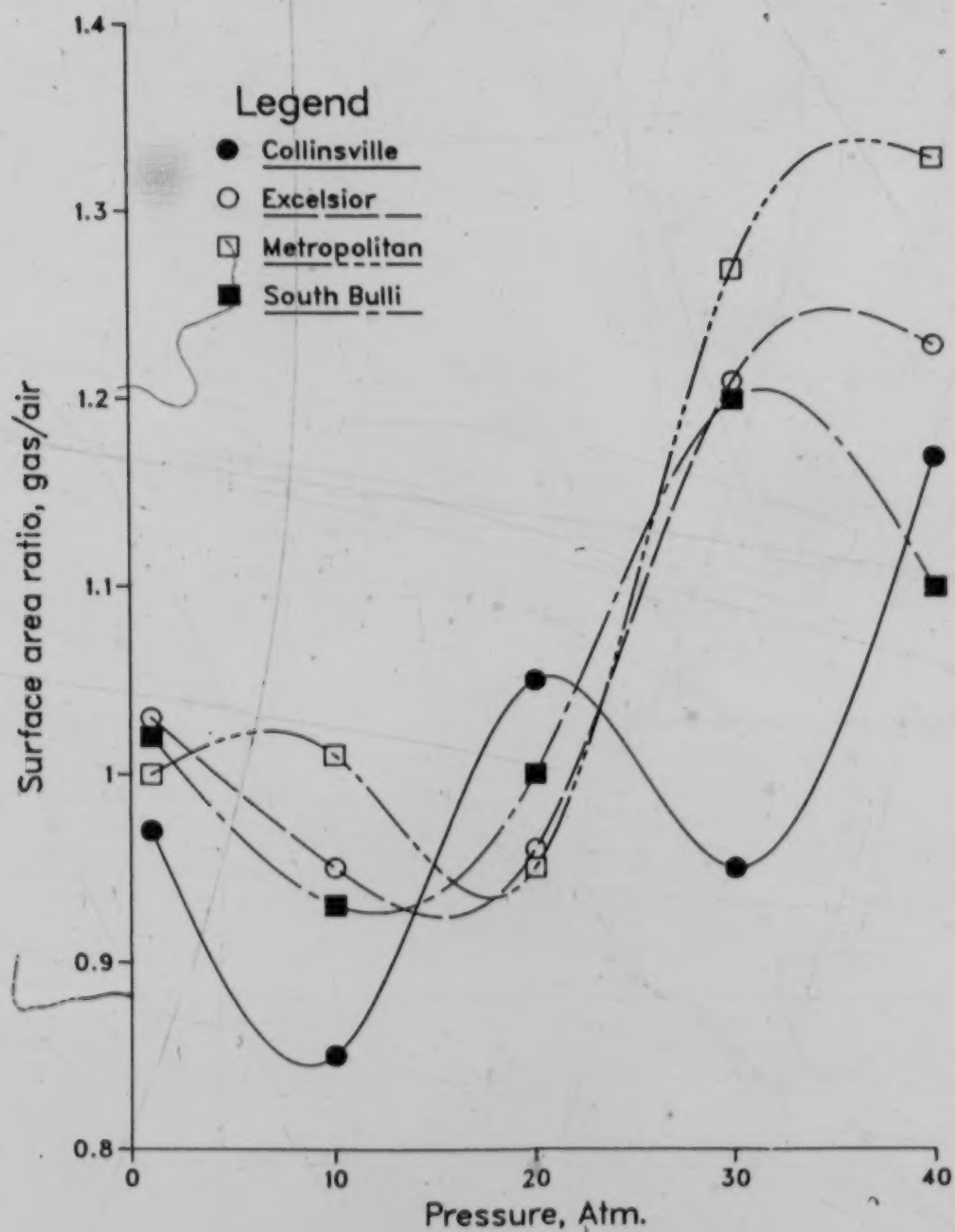


Figure 11: Surface area ratio($\Delta S_{CO_2}/\Delta S_{air}$) vs. pressure.

(Data from Tankard, 1957)

Conclusions derived: One of the basic conclusion derived by Tankard is that:

"...The drop weight strength of four coals examined, is significantly affected by sorption of carbon dioxide....At 10 atmospheres the coal is significantly stronger than when tested in air; from 10-35 atmospheres coal loses strength, and at 40 atmospheres it has regained some of the lost strength." The reason for this phenomenon is explained as :

"Strengthening at 10 atm. is quite marked particularly for the Collinsville coal where a 15% increase over the air value was obtained. ...At pressures up to 10 atmospheres , the sorbate molecules are mainly adsorbed on the macro pore surface. A few however, interpose themselves between lamellae lending a certain flexibility to the coal structure. This flexibility, or elasticity, results in lower ΔS values. Above 10 atmospheres the micro pores become accessible to the gas molecules....The pressure gas in the micro pores stresses the coal structure, while interwedged molecules decrease the bonding between lamellae. The coal loses its flexibility, becomes more brittle and as a result increasing ΔS values are obtained."

Analysis:

Tankard's experiment results are well documented in the work referred. A t-test (see Appendix 15), at 95% level of confidence, on the relevant data shows that there is no significant difference between the mean strength values obtained by testing coal at 1 atm. and at 20 atm. in carbon

dioxide gas, for all the types of coals. This means that there is no significant strengthening (contrary what was concluded) or weakening of coal at this pressure range, and that the difference in mean values is due to variation of the individual results. There is, however a significant difference between the strength values obtained at 20 atm. and 40 atm. The reason for this difference can not necessarily be attributed to the adsorption, for the reasons explained in the following:

1. The experimental method is such that the coal is subjected to pressure, and after 20 min. sorption period it is de-pressurized. In pressurizing and in particular releasing the gas during the transfer from the sorption bomb to crusher, it is likely that coal is damaged, if not subjected to a tensile failure.
2. Among the four coal types that tested only Collinsville coal shows a strengthening when tested with 10 atm. CO₂ as compared to testing in air. Therefore, a general conclusion can not be derived.
3. The explanation that in the micropores the molecules of the gas interpose themselves between lamellae lending a flexibility to coal structure, thus making it stronger, is questionable in the sense that if true, the same effect should apply in the micro pore region as well. Furthermore, this "flexibility" did not occur with other types of coals.

Therefore, the procedure as followed by Tankard is not satisfactory. The results obtained do not show a strengthening (nor weakening) in the test ranges of 1-20 atm. pressure. There is a weakening of coal between 20-30 atm. range. Because the experiments were not repeated with a non adsorbing gas, the reason for this weakening is not clear and cannot necessarily be attributed solely to adsorption.

In summary, the effect of sorption of mine gases on the strength of coal in particular is not clear from the studies mentioned. The studies of Desai and Gregg involve using coal compacts with fluids other than mine gases. Ettinger used an indirect approach. Tankard's experimental method and interpretation of his results are questionable. Furthermore there is no agreement on whether the sorption of gases reduces the strength of solids, such as expressed by Sereda and Feldman (1967): "...Physical adsorption plays a minor role, if any, in the strength reduction of solids."

TABLE 3
PREVIOUS INVESTIGATIONS

PARAMETER STUDIED	MATERIAL TESTED	FLUID USED	RESULTS	REFERENCE
Strength	Glass slides	Water vapour	Large decrease in bending strength at low humidities but no further change when going to saturation.	Shoening F.R.L., J. of Appl. Phys. 31,(1960)
Strength	Silica rods	Vapours of water, ethyl alcohol, n-buthyl alcohol, ethyl acetate.	Water, even when present in trace amounts (0.02% in benzene) lowers the strength considerably more than do others.	Hammod and Ravitz J. of Am. Ceram. Soc. ,46(1963)

TABLE 3(cont'd)

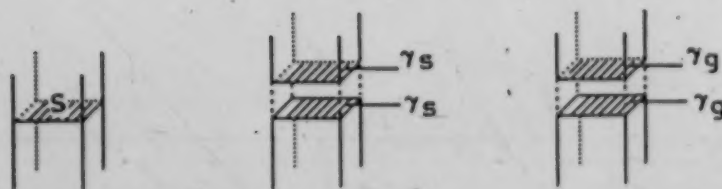
PARAMETER STUDIED	MATERIAL TESTED	FLUID USED	RESULTS	REFERENCE
Strength	Glass	Various vapours	The strength of ordinary soda glass fell from 172 Mpa in vacuum to 83 Mpa in saturated water vapour.	Baker T.C. and et al, J. of Appl. Phys. 17(1946)
Strength	Compacts of kaolin	Water vapour	The strength of kaolin measured in presence of water vapour diminished regularly with increase in relative pressure. It was 1/5th of vacuum value.	Dolimore and Gregg Trans. Brit. Ceram. Soc. 54(1955)

4.3 THEORETICAL CONSIDERATIONS

4.3.1 REDUCTION IN THE SURFACE ENERGY WITH ADSORPTION

The specific energy of a solid is the isothermal reversible work required to be done in creating 1 cm^2 of a new surface. The amount of such work is less in cases where adsorption is effective on the surface of the solid. The reduction in surface energy of a solid brought about with adsorption can be illustrated by the following example (adapted from Gregg 1961):

Consider an imaginary experiment in which a prism of solid, 1 cm^2 in crosssection is pulled apart:



If the experiment is conducted in vacuo, the required energy will be $2\gamma_s$, and in the thermodynamic sense it will be a free surface energy rather than a total surface energy (surface enthalpy). The converse situation would be restoring the prism in its original condition, and the

energy gained would be the same as in pull apart situation; i.e. $2\gamma_s$. If, the same experiment were performed in presence of gas or vapour which had formed an adsorbed film on the surfaces, the gain in the energy, $2\gamma_g$, would be less than $2\gamma_s$, because the force of attraction between two solid surfaces has now in effect been replaced by that between two adsorbed films which have a weaker Van der Waals field than the solid.

The reduction $(\gamma_s - \gamma_g)$ or simply $(\gamma_0 - \gamma)$ in the free energy caused by adsorption of a film in equilibrium with a gas at pressure P , may be calculated by the Gibbs adsorption isotherm as follows:

$$\Delta\gamma = \gamma_0 - \gamma$$

$$-d\gamma = RT\Gamma d(\ln P)$$

Where R is the gas constant, T is the absolute temperature, and Γ is the number of moles adsorbed per cm^2 when the pressure of gas is P .

By integration from zero pressure (where $\gamma = \gamma_0$) to pressure P we obtain

$$\gamma_0 - \gamma = RT \int_0^P \Gamma d(\ln P)$$

$$\text{Since } \Gamma = \frac{x}{MS}$$

(Where x is the number of grams adsorbed per gram of solid,

M is molecular weight of gas and S is the surface area of the solid),

$$\Delta\gamma = \frac{RT}{MS} \int_0^P x d(\ln P)$$

The above equation can be evaluated from the experimental adsorption isotherm, if the surface area, S, is known.

4.3.2 THEORETICAL STRENGTH OF AN ELASTIC MATERIAL

According to Orowan (1949) the molecular strength, σ_0 of an elastic material is given by:

$$\sigma_0 = \sqrt{\frac{2\gamma E}{a}}$$

Where σ_0 is the magnitude of uniaxial tensile stress required to overcome the molecular cohesion of the material, γ is the specific surface energy of the material, E is the Young's modulus, and a is the spacing between neighbouring atomic planes.

However, in general the materials fail at a stress level lower than that predicted by the above equation. Griffith's theory of brittle fracture initiation is generally regarded as an acceptable explanation for the discrepancy between the observed and theoretical strengths of brittle materials.

GRIFFITH THEORY OF BRITTLE FRACTURE INITIATION

Griffith's theory assumes the presence of flaws or microcracks in the materials. When the material is stressed, high stress concentrations occur around the "favorably" oriented cracks which will lead to a lower strength.

By assuming that a crack in a two dimensional body can be approximated by an ellipse of large eccentricity, Griffith was able to calculate, from analytical solution of stress distribution around elliptical openings in a plane body under stress, that maximum stress at the end of a crack oriented at right angles to the direction of applied tension is given by

$$\sigma_0 = 2\sigma_a \sqrt{\frac{c}{r}}$$

Where:

σ_a is the applied stress,

$2c$ is the length of the crack, and

r is the radius of the end of the crack.

Griffith based his calculations upon assumptions concerning the energy required to propagate the crack. Considering the energy changes associated with the crack propagation, and that the energy balance of the system must be maintained, it can be shown (see Appendix 3) that the condition for crack propagation under plane stress is given by:

$$\sigma_a = \sqrt{\frac{2\gamma E}{\pi C}} = \sigma_t \dots \dots \dots (1)$$

Where σ_t is the tensile strength of the material.

4.3.3 FRACTURE STRESS IN A THREE DIMENSIONAL SYSTEM

Griffith's original work considers a two dimensional crack. Because the actual cracks are three dimensional in shape, and are acted upon by three principal stresses, a three dimensional treatment of the problem was necessary. Sack (1946) has extended Griffiths's work to the case of a crack which has the shape of very flat oblate ellipsoid ("penny shaped" crack). He has shown that the most dangerous cracks lie in plane parallel to the direction of the intermediate principal stress which has negligible influence upon the stress distribution at the crack tip. The magnitude of the applied stress required to cause propagation of the particular crack was found to be :

$$\sigma_a = \sqrt{\frac{\pi \gamma E}{2C(1-\nu^2)}}.$$

4.3.4 FRACTURE STRESS UNDER UNIAXIAL TENSILE STRESS CONDITIONS

In deriving the conditions for fracture initiation in a brittle material, the uniaxial strength (σ_t) has been chosen as a basic material constant. The reason for this choice is

that the uniaxial tensile strength is theoretically related to the molecular cohesive strength σ_0 , which assumed to govern the mechanism of brittle fracture initiation by:

$$\sigma_0 \xi_0 = 2\sigma_t$$

Where ξ_0 is the eccentricity of the ellipse.

The molecular strength, σ_0 , and crack geometry, ξ_0 , can not be determined by direct physical measurements. However it is shown by Hoek (1965) that when the principal stress ratio (σ_3/σ_1) , ≤ -0.33 , then,

$$\sigma_0 \xi_0 = 2\sigma_3 \text{ i.e}$$

$$\sigma_3 = \sigma_t.$$

Where σ_t is given by equation (1). (Page 65).

4.4 REDUCTION IN THE STRENGTH OF BRITTLE MATERIALS DUE TO ADSORPTION

4.4.1 UNIAXIAL TENSILE STRESS CONDITIONS

If we compare the tensile strength of the same material without the adsorption effect, σ_{t0} , and with the adsorption σ_t :

$$\sigma_{t0} = \sqrt{\frac{2\gamma_0 E}{\pi c}}$$

$$\sigma_t = \sqrt{\frac{2\gamma E}{\pi c}}$$

and assuming that Young's modulus, E , of the material remains unchanged *

$$\frac{\sigma_t}{\sigma_{t0}} = \sqrt{\frac{\gamma}{\gamma_0}}$$

Assuming the initial surface energy (γ_0) as unity,

$$\gamma_0 - \gamma = \Delta\gamma.$$

$$\gamma = 1 - \Delta\gamma \text{ and,}$$

$$\frac{\sigma_t}{\sigma_{t0}} = \sqrt{1 - \Delta\gamma}.$$

*This assumption may not be correct. Although Shoening (1963) showed that Young's modulus is not affected by the adsorbed water vapour, it is difficult to envisage no change in Young's modulus when adsorption is accompanied by swelling (as is the case with CO_2 adsorption).

$\Delta\gamma$ can be calculated from the experimental adsorption isotherm with the application of the Gibbs equation (see Section 4.3.1). Using the adsorption isotherm given in Figure 4 (page 25) for low volatile bituminous coal, it is calculated that (see example, Section 4.4.4) a 10% reduction in the strength of coal may be expected as a result of methane adsorption on coal.

4.4.2 TRIAXIAL STRESS CONDITIONS

For the triaxial stress conditions the fracture initiation criterion can be expressed by the following equation (Hoek, 1965)

$$\sigma_1 = \frac{-8\sigma_t(1 + \kappa)}{(1 - \kappa)^2}$$

Where

σ_1 is the major principal stress at fracture

σ_3 is the minor principal stress at fracture

$$\kappa = \frac{\sigma_3}{\sigma_1} \geq -0.33$$

or

$$\sigma_1 = \frac{-8(1+\kappa)}{(1-\kappa)^2} \sqrt{\frac{2\gamma E}{\pi 2c}}$$

Thus by the previous analogy,

$$\frac{\sigma_{10}}{\sigma_1} = \sqrt{1 - \Delta\gamma}.$$

Where σ_{10} is the magnitude of major principal stress at fracture without the effect of adsorption, and σ_1 is the magnitude of major principal stress at fracture with the effect of adsorption.*

4.4.3 ADSORPTION AND FRACTURE PROPAGATION

It is already established, on the basis of Griffith's equation and Gibbs adsorption equations, that a reduction in the surface energy occurs as a result of adsorption and that this causes a reduction in the tensile strength which must be overcome at the tip of the Griffith's crack. This is the equilibrium micro-crack situation. A further incremental stress will cause the microcracks to propagate thus creating a macro fracture system on the surfaces of which spontaneous adsorption continues. In this manner the surface energy of the solid is reduced.

4.4.4 EXAMPLE CALCULATION

Using the adsorption isotherm, for the low volatile bituminous coal, given in the Figure 4 (page 25), the following data can be obtained:

*An alternative approach to considering the effects of adsorption under triaxial stress conditions would be to consider the shear strength of the material. It would be anticipated that the change of surface energy would reduce both the cohesion and angle of internal friction of coal.

Pressure(Atm.)	10	20	30	40	50
Pressure(kPa.)	981	1962	2943	3926	4905
$\text{Log}_{10}P(\text{Pa})$	5.99	6.29	6.47	6.59	6.69
Adsorption $\frac{\text{m}^3 \text{methane at STP}}{\text{tonne coal}}$	18.6	22.1	24.6	26.6	28.0

To obtain the amount of methane adsorbed in g/g-coal,

$$PV = nRT$$

Where

P is the pressure in Pascals

V is the volume at STP, in m^3

n is the number of moles

R is the gas constant, 8.314 Joules/K.mol

T is the temperature, in degrees Kelvin.

$$n = \frac{PV}{RT}$$

$$n = \frac{(98.1 \times 10^3)(18.6)}{8.314 \times 273} = 803.9 \text{ moles}$$

$$803.9 \text{ moles} \times 16 \text{ g/mole (mol. wt. of methane)} = 12,862$$

$$\text{g/tonne coal} = 0.013 \text{ g/g coal} = x,$$

Similarly at the other pressures;

$$x_2 = 0.015 \text{ g/g coal.}$$

$$x_3 = 0.017 \text{ g/g coal.}$$

$$x_4 = 0.018 \text{ g/g coal.}$$

$$x_5 = 0.019 \text{ g/g coal.}$$

since

$$\Delta\gamma S = \frac{RT}{M} \int_0^P x d(\ln P),$$

The integral part can be obtained graphically by plotting x (The amount of the adsorption, g/g coal) vs. $\log P$ as shown in the Figure 12 (Page 74). Thus, for this example the value of $\int_0^P x d(\ln P) = 0.035 \times 2.303 = 0.08$, and

$$\Delta\gamma S = \frac{8.314 \times 273}{16} \times 0.08 = 11.43 \text{ Joules m}^2$$

The value of $\Delta\gamma$, i.e the amount of reduction in the surface energy due to adsorption, can be found if the surface area of this coal is known. This information is not available. However, to continue the example, the surface area corresponding to medium volatile bituminous coal measured at 0 °C can be obtained from Table 1 (page 12),

being as $62 \text{ m}^2/\text{g}$. Then:

$\Delta\gamma$ will be 0.184 Joules/gr .

According to the Griffith's theory the tensile strength of a solid material can be obtained by the following equation:

$$\sigma_t = \sqrt{\frac{2\gamma E}{\pi c}}$$

Where,

σ_t is the tensile magnitude of the uniaxial tensile strength of material,

γ is the specific surface energy of the material,

E is Young's modulus, and

$2c$ is the crack length.

Now, if we denote σ_{t0} as the tensile strength when no adsorption effect is present, and σ_t when there is adsorption and reduction in the surface energy,

$$\sigma_{t0} = \sqrt{\gamma_0 a}$$

$$\sigma_t = \sqrt{\gamma a}$$

Where a is a constant (assuming all other parameters remain the same in both cases). Then,

$$\frac{\sigma_{t0}}{\sigma_t} = \sqrt{\frac{\gamma_0}{\gamma}}$$

Assuming $\gamma_0 = 1$,

$\gamma = 1 - \Delta\gamma$, and

$$\frac{\sigma_{to}}{\sigma_t} = \frac{1}{\sqrt{1 - \Delta\gamma}} = \frac{1}{\sqrt{1 - 0.18}} = \frac{1}{\sqrt{0.82}} = \frac{1}{0.9}$$

Therefore $\sigma_t = 0.9\sigma_{to}$

Thus a 10% strength reduction should be expected as a result of adsorption of methane on coal for this particular case. Since the amount of adsorption of CO_2 on coal is almost twice as that of CH_4 (see Section 2.2.7), the strength reduction with CO_2 can be as high as 20%. Also it was assumed that the Young's modulus, E , does not change with the adsorption of the gas on coal. Since adsorption is accompanied by swelling, this assumption may not be applicable. A reduction in Young's modulus will result even higher reduction in strength.

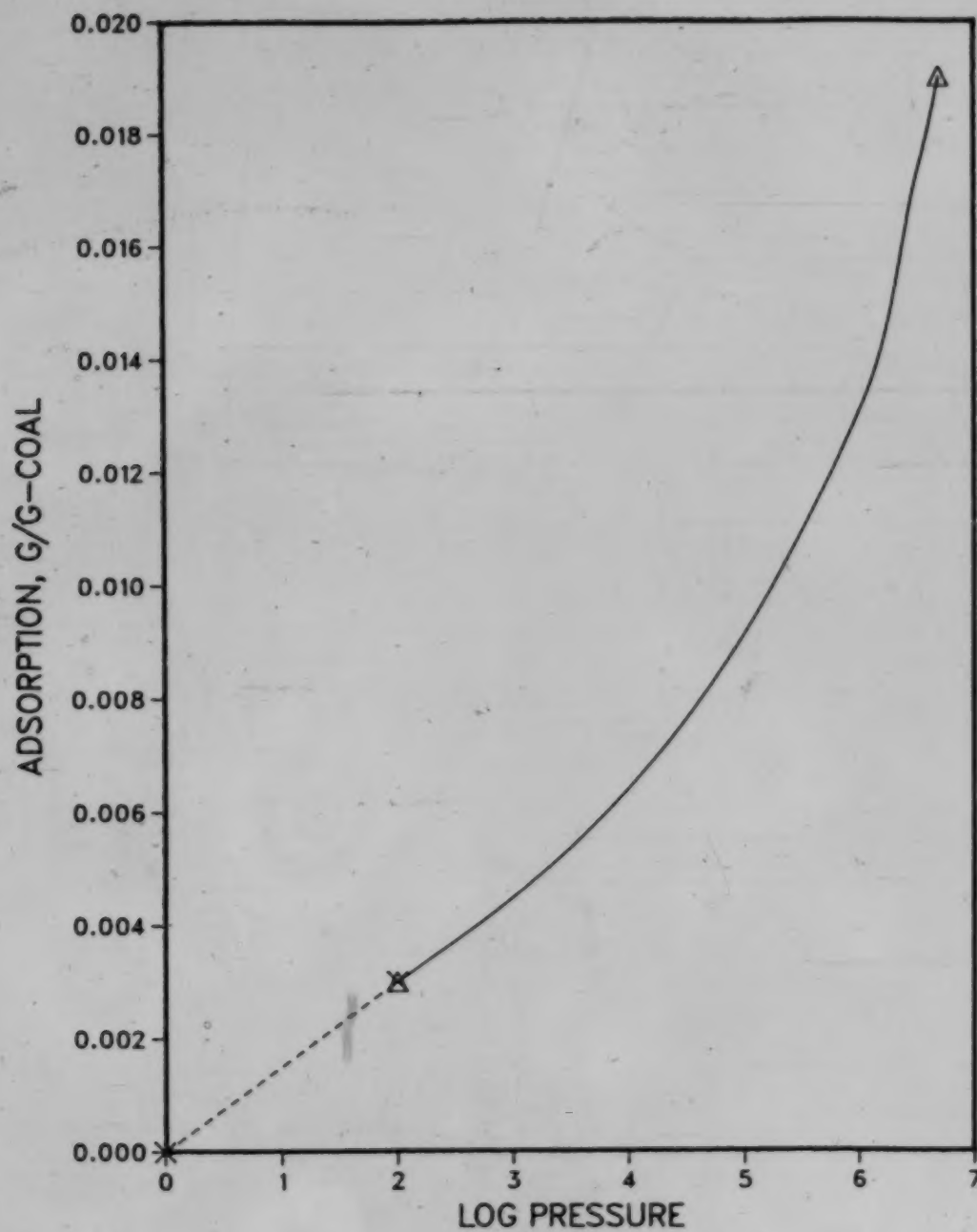


Figure 12: Adsorption(g/g-coal) vs. Log_{10} pressure.

5. CLASSIFICATION OF COALS TO BE TESTED

Two types of coals were used in the experiments: coal from Highvale open pit mine in Alberta and from Metropolitan Colliery in Australia. Highvale coal was the major source for the tensile strength tests. A limited number of disc samples from the Metropolitan coal were also prepared for testing at a later date.

Samples from both types of coals were tested for their moisture content, ash content, and volatile matter content. Their elemental carbon and hydrogen contents were also determined. The method followed in these tests is based on ASTM procedure (ASTM D388-77, part 26). Results of these tests are as follows:

ANALYSIS	COAL TYPE	
	Highvale	Metropolitan
Moisture, %	15.90	0.79
Ash, %	12.54	9.20
Volatile matter, %	28.25	22.54
Fixed carbon, %	43.31	67.47
Fixed carbon(dmmf), %	61.60	75.73
Carbon(daf), %	75.46	86.32
Hydrogen, %	4.48	4.13
Rank	Sub Bit.	Med. Vol. Bit.

ΔP INDEX:

The ΔP index, for classification of coals according to their degree of danger to outbursting, was developed in Russia by Ettinger and later was modified by the French institution known as CERCHAR. The apparatus and detailed description of the test procedure is given by Singh, 1968. Briefly, the test procedure is as follows:

1. A 3 gm. sample of -32/+60 mesh size is placed in a cup, evacuated, and charged with helium at 1 atm. for 10 minutes. The sample is then connected to a manometer and the pressure (P_1) is recorded. The value of P_1 is a measure of the contribution of free gas which is not sorbed by the sample but held under atmospheric pressure in the coal pores and intergranular spaces.
2. The helium is now evacuated from the sample (vacuum re-established) and methane is introduced for 1 1/2 hours.
3. After the sample is saturated, it is re-connected to the manometer and the pressure (P_2) is recorded after a 60 second desorption period.
4. The ΔP index is calculated from :

$$\Delta P = P_2 - P_1$$

Attempts have been made to relate ΔP index and degree of danger to outbursting; with limited success. For example, Vandeloise (1964) reports the classification of ΔP index values applicable to Belgian coals with outburst of methane

as :

ΔP_{0-60} (mm. Hg) DEGREE OF DANGER

00 - 15	Coal not prone to outburst.
15 - 30	Slightly suspected coal.
30 - 45	Suspected coal.
45 - 60	Dangerous coal.
>> 60	Highly dangerous coal.

Both types of coals used in this study were tested for their respective ΔP values. Results are as follows:

Highvale: 21.0, 21.7, 21.0, 21.8, 20.5, 20.5 (mm. Hg)

Average = 21.1 mm. Hg

Metropolitan: 24.2, 23.7, 23.4, 23.0, 23.0, 22.9.

Average = 23.4 mm. Hg

The ΔP values obtained for the Highvale and for the Metropolitan coals are quite low. Although the ΔP value for the Metropolitan coal is found to be higher in all samples tested, it would still be in "slightly suspected coals" category. On the other hand, outbursts have been occurring in the Metropolitan Colliery.

It should be noted that the samples were tested after a considerable lapse of time and had been exposed to

atmosphere. The oxidation of the samples will result lower ΔP values. As much as 25% drop in ΔP is recorded over a period of one year depending on the storage conditions (Lama et al 1983).

While the oxidation of the samples may be the reason for the low ΔP value for the Metropolitan coal, the ΔP method was found not to be a reliable "outburst index" in Australian coal mines (Hargraves, 1983).

6. EXPERIMENTAL EQUIPMENT AND PROCEDURE

6.1 THE BRAZILIAN STRENGTH TEST

The Brazilian strength test is an indirect method used for determining the tensile strength of brittle materials. In this test a rock disk of diameter D and thickness t is loaded along its diameter with a load P . The sample usually fails by splitting along the line of diametral loading.

It is generally accepted that the failure occurs as a result of uniform tensile stress normal to the loading plane and the tensile strength, σ_t , of rock is calculated from the formula:

$$\sigma_t = \frac{-2P_{\max}}{\pi D t}$$

THE VALIDITY OF THE BRAZILIAN TEST

The theoretical principal stress equations, for the loading conditions in the Brazilian test, are given by Frocht (1947) as:

$$\sigma_1 = \frac{-P}{\pi r t} \cdot \left(\frac{Z_1 Z_2 + 2r^2 Z_2}{Z_1 Z_2 + 2r^2 Z_3} \right)$$

$$\sigma_3 = \frac{-P}{\pi r t} \cdot \left(\frac{Z_1 Z_2 - 2r^2 Z_2}{Z_1 Z_2 + 2r^2 Z_3} \right)$$

Where, with the center of the coordinate system being at the center of the disc,

$$Z_1 = x^2 + y^2 + r^2,$$

$$Z_2 = x^2 + y^2 - r^2,$$

$$Z_3 = x^2 - y^2 + r^2,$$

r = Radius of the disc.

The stress distribution given by the above equations, as plotted by Colback (1966), is given in Figure 13. Also shown in this figure is the stress distribution along the loaded diameter of the disk.

Hoek (1965), based on the analysis of the Griffith's Theory, gives the conditions for the tensile fracture ($\sigma_3 = \sigma_t$) as $\frac{\sigma_3}{\sigma_1} = \kappa \leq -0.33$ and $\psi=0$, ψ being the angle between the major principal stress direction and the critical crack (from which the fracture initiates) orientation. From Figure 13 it can be seen that only at the center of the disc the principal stress ratio, κ , satisfies the $\frac{\sigma_3}{\sigma_1} = -0.33$ thus $\sigma_3 = \sigma_t$ condition. From this consideration and from the plots of the critical crack orientations, Colback concluded that in order to consider the test a valid one, the fracture initiation must start from the center of the disc.

In practice the determination of this condition, i.e whether the fracture initiated at the center of the disc or not, is quite difficult and the presence of a diametrical fracture plane is sufficient to accept the test as a valid one.

Several types of failures were observed during the tests performed for the present study (Figure 14). Of the failure types shown in this figure only the types c, and d

were considered to have occurred as a result of tensile failure.

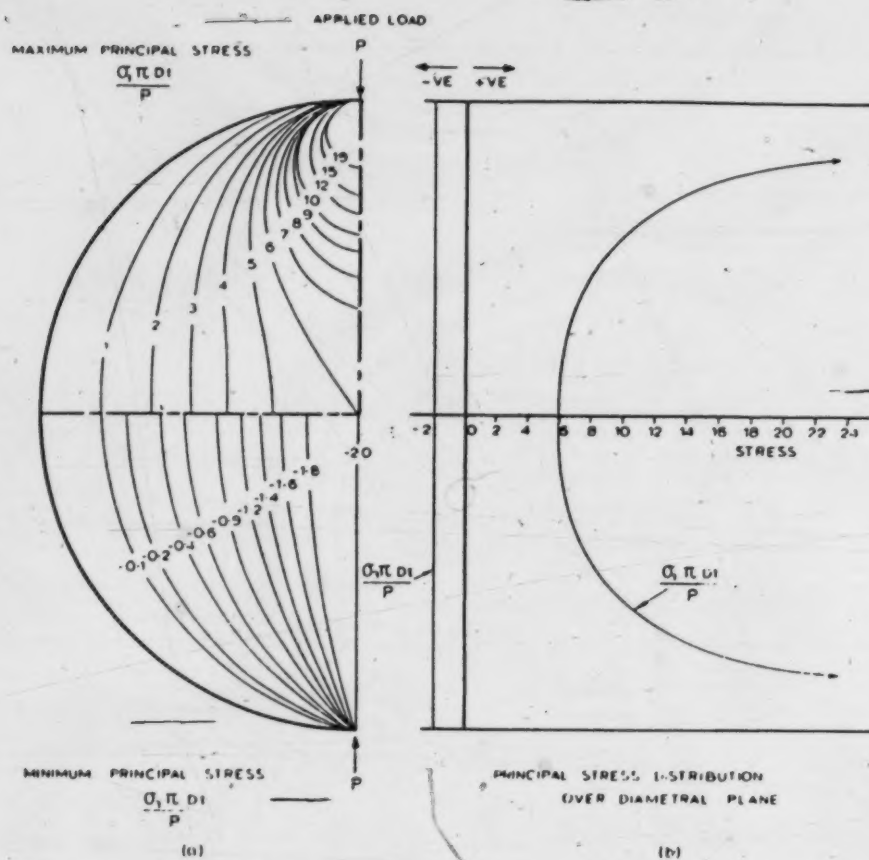


Figure 13: Theoretical stress distribution in a disc subjected to a diametral compressive point load P . (After Colback, 1966).

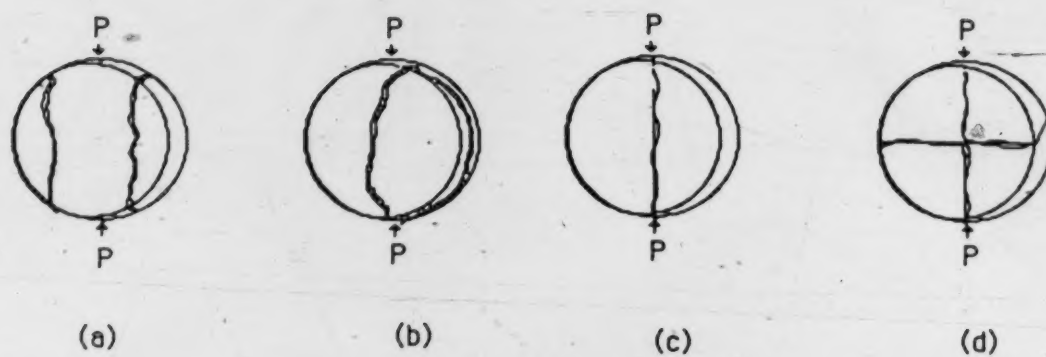


Figure 14: Observed failure types with the application of a compressive point load on disk coal specimens.

6.2 APPARATUS

A testing apparatus was designed and built to serve as a test facility in which Brazilian indirect tensile strength tests can be carried out under gas pressure. The details of the apparatus are shown in figures 15 - 17. It consists of a cylinder and a sample holding unit that is attached to the top lid of the cylinder. Also attached to the top lid of the cylinder is a hydraulic ram which will provide the necessary load during testing. Two rubber "O rings" placed at the top and at the bottom of the cylinder walls provided the necessary leak proof conditions. The top cap design allows the top lid, and the sample holding unit attached to it, to be lifted in and out of the cylinder independent of the top cap. The sample holding unit consists of three threaded bars, a steel platform and an adjustable screw plate. This design allows the circular disk sample to be secured between the hydraulic ram and the adjustable screw plate. Other equipment used with the apparatus are a hydraulic pump connected to the hydraulic cylinder, a vacuum pump, a pressure gauge (0 - 6.9 Mpa), a gas cylinder with the necessary pressure regulators, pressure gauges, valves, and tubing.

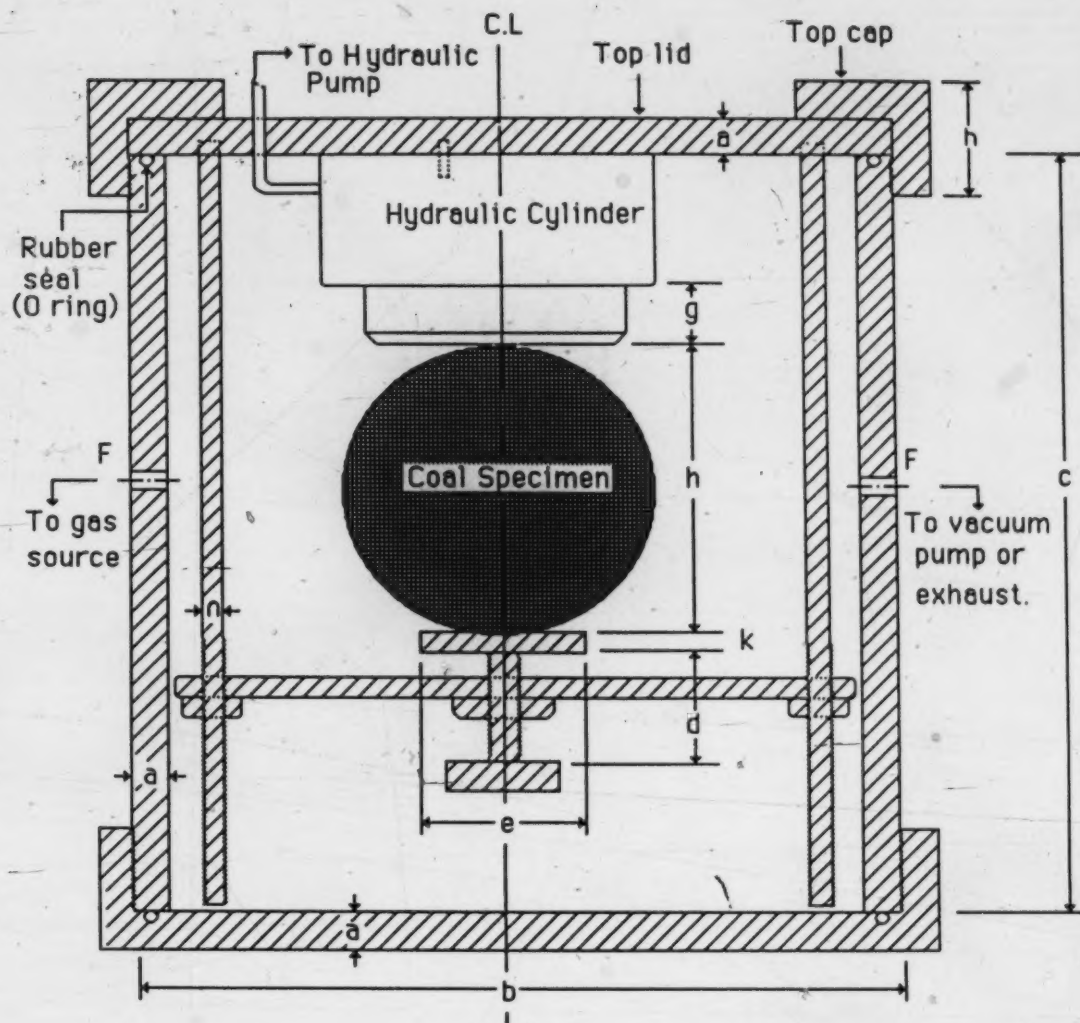


Figure 15: Testing Apparatus (not true scale)

a = 19.1mm b = 203.2mm c = 228.6mm

d = 25.4mm e = 38.1mm g = 6.4mm

h = 54.1mm k = 6.4mm

F : 6.4mm through hole with 3.3mm NTP to 12.2mm depth.

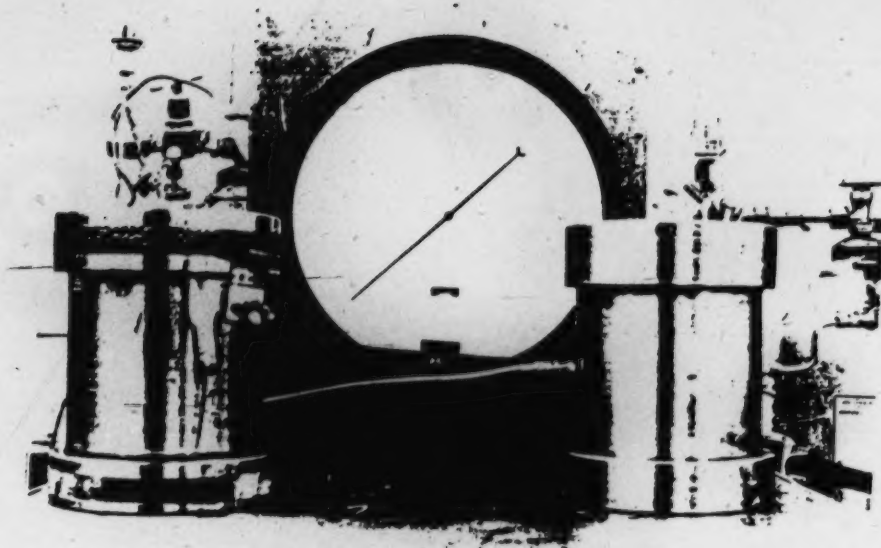


Figure 16: A picture of experiment set-up.

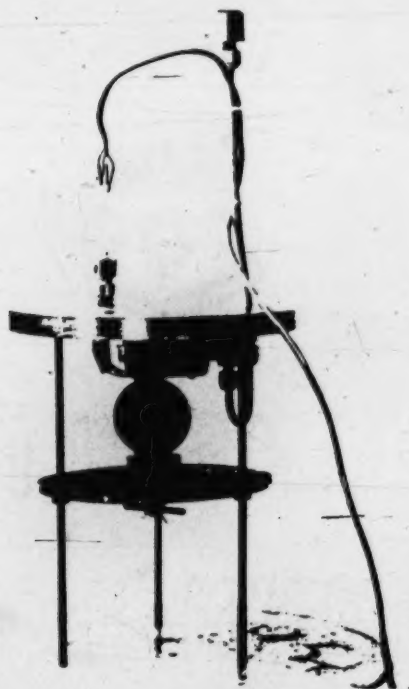


Figure 17: Sample holding unit.

6.2.1 CALIBRATION

The apparatus involves two pressure gauges: A hydraulic gauge for measuring the pressure applied to the sample through the ram, and a gas pressure gauge measuring the gas pressure inside the cylinder. The gas pressure gauge was calibrated against two other calibrated gauges and shown to give correct readings. The hydraulic gauge was calibrated against the gas pressure gauge using the following procedure:

1. A proving ring was designed and calibrated against a commercial proving ring for its load measurements.
2. The calibrated proving ring was placed into the sample holding unit (where normally a coal disc specimen would be placed), and the output was connected to a chart recorder through the top lid of the cylinder.
3. The cylinder was evacuated (as it would be during the actual testing), and then pressurised with gas to a desired level. At this pressure level the load was applied by the hydraulic pump until the chart recorder started to indicate a reading. Since at this point the hydraulic ram is under equilibrium pressure, the readings from two pressure gauges should be the same.
4. Step 3 was repeated several times and for different gas pressure levels. Figure 18 shows the calibration so obtained. From the comparison of two gauge readings, the following regression equation was obtained:

$$Y = 105.09 + 0.826 X \text{ (cor.(X,Y) = 0.99993)}$$

Where Y is the reading at the hydraulic gauge, and X is the reading at the gas pressure gauge.

It is seen that the hydraulic pressure gauge applying pressure via the ram responds with a constant error, probably due to friction in the piston. Consequently the readings were corrected in accordance with the above equation to obtain the correct load measurements.

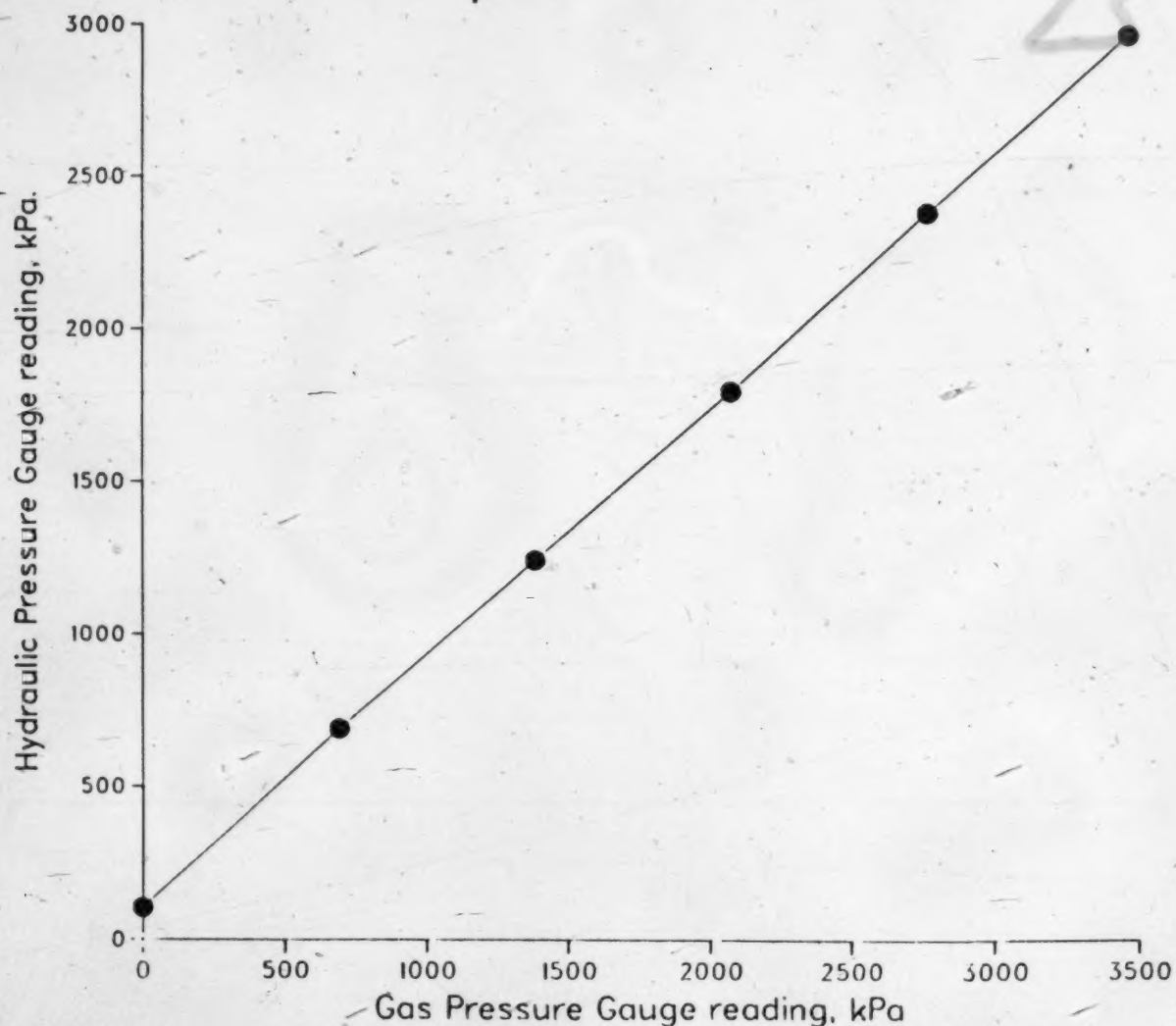


Figure 18: Calibration Chart.

6.3 EXPERIMENTAL PROCEDURE

6.3.1 SAMPLE PREPARATION

The coal samples in the form of blocks were taken from Highvale mine, Pit O2S Cut 14, on July 10 1986. These coal blocks were packed and sealed in plastic bags and stored in a cooler at approximately 5°C. They were cored with a NX(54mm) size core bit between July 12 and July 30. The individual disk samples with approx. 12.7mm (0.5in) thickness were also cut from the cores during that time. Each disc was marked according to the block and the core it came from as shown in Figure 18. For example, the sample 1-1-3 is the third disc from the first core of the first block of coal. The date the sample was taken was also recorded.

A set of random numbers were generated and the sample numbers were replaced with these random numbers in the absence of the author. Then, the samples were re-grouped to be tested for the desired test conditions. (Example : re-grouping according to the last digit in sample random number: Prepare a group with the last digit being 5. If an insufficiently large group is obtained, choose the rest from others with last digit being 6.) This procedure allowed random testing of the samples. Thus, it was ensured that samples chosen to be tested under different test conditions (pressure, gas type, etc.) represent the same sample population, i.e all the coal blocks, and that statistical

6.3.2 TEST PROCEDURE

The sample to be tested was placed in the holding unit as shown in the Figure 17. The holding unit then was placed in the main cylinder and evacuated for half hour. For the vacuum tests the sample was tested after evacuation for 30 minutes. For the tests involving gas, the outlet valve is closed immediately after the half hour evacuation period and the inlet valve was connected to the gas cylinder for the desired gas pressure level. The gas inlet valve was then closed and the sample was left under this pressure for the established sorption time, which for CO₂ gas was established as being 24 hours (see below). The sample was tested for its tensile strength after 24 hours.

Originally a proving ring was designed, calibrated, and used in connection with a XY plotter in order to record the maximum load during testing. However, difficulties were experienced due to strain gauge "pop-outs" during subsequent evacuation and loading. This may have been caused by the inadequate bond type used, or simply due to "aged" bonding cement used. During the calibration of the proving ring it was observed that a hydraulic pressure gauge with the maximum pressure indicator on it would be sufficiently accurate, provided that its readings were corrected in accordance with the calibration curve given in section 6.2.1.

The sorption time of 24 hours was established on the basis of preliminary desorption tests. In these tests, the

— samples were left in gas pressures for long periods of time (36-48 hrs.) and then the pressure was reduced to atmospheric pressure (at an approximate rate of 0.7 Mpa/min.). Then, the outlet pipe was connected to a manometer. The time after which the pressure stabilized was taken as the critical time which is required for sorption to be completed. This time, for CO_2 , was found to be 17 hours (Figure 20). For practical reasons all the samples tested for uniaxial tensile strength involving CO_2 , were left under the pressure for 24 hours.

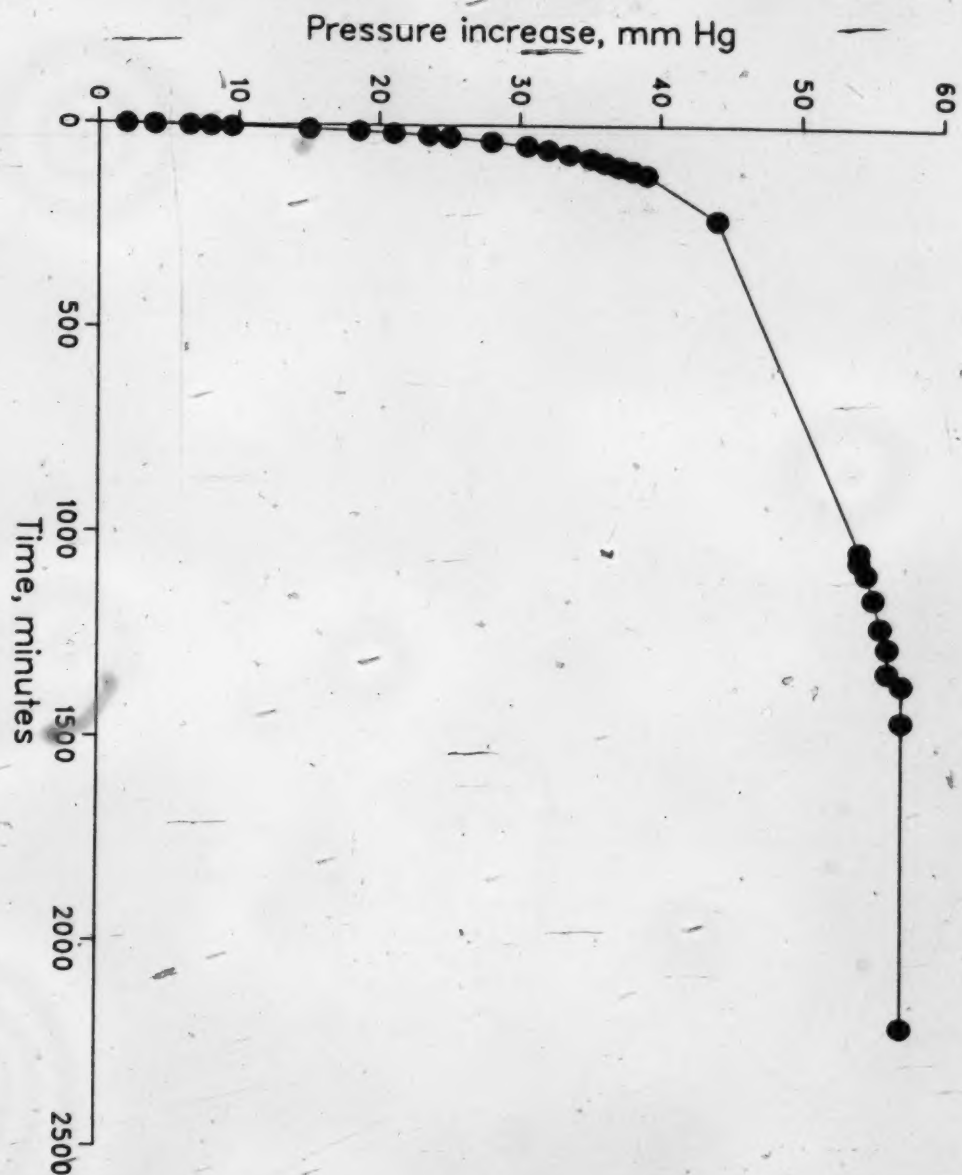


Figure 20: Desorption time vs. pressure.

7. PRELIMINARY TEST RESULTS AND EVALUATION

7.1 RESULTS

By using the apparatus and the method described in the previous chapter, the coal samples from Higvale mine were tested for their tensile strength under conditions of :

- a) Vacuum
- b) Saturated with He gas at 3.45 Mpa, and
- c) Saturated with CO₂ gas at 3.45 Mpa.

With anticipation that :

- Vacuum results will serve as a basis for comparison.
- By testing coal samples saturated with Helium (non-adsorbing gas), it will be possible to compare with the results obtained under vacuum condition for identification of differences due only to pressure (if any), and further to compare the results with those obtained by the saturation with CO₂ (adsorbing gas).
- By comparing the results obtained with saturation by CO₂ and those of He, the effect of adsorption on the strength of coal will be determined.

The individual results of these tests are given in Appendices 4,5 and 6, and summarised in Table 4.

TABLE 4
SUMMARY OF THE PRELIMINARY TEST RESULTS

TEST CONDITION	NO. OF SAMPLES TESTED	TENSILE STRENGTH (kPa)	
		MEAN	ST. DEVIATION
Vacuum	21	855.2	179.2
Saturated with He at 3.45Mpa	10	927.5	176.6
Saturated with CO ₂ at 3.45Mpa	20	793.4	144.2

7.2 EVALUATION OF THE PRELIMINARY TEST RESULTS

The results of three sets of experiments (vacuum, CO₂ at 3.45 Mpa, and Helium at 3.45 Mpa) show that:

a) There is no significant difference, at 95% confidence level, between the Helium results at 3.45 Mpa and those in vacuum.

b) There is a 0.13 Mpa (14%) reduction in the mean strength of coal when tested under CO₂ gas pressure as opposed to testing under He gas pressure. The difference, which is significant at 95% level of confidence (t - test), may be due to sorption effect of CO₂ on the strength of coal. However there is also a possibility that the specimens may have been "damaged" due to relatively fast injection of the gas in to the apparatus as explained below.

During the pressure tests, the sample was evacuated first and subsequently was subjected to a 3.45 Mpa gas pressure. If the penetration of gas into the specimen is time dependant, then the high pressure around the sample can cause the sample to compact, and as a result the collapse of pore structure may occur. If such a pressure gradient was created due to the experimental method, then there could be damage to the specimen resulting in a lower strength than otherwise would be expected. Because He has a smaller molecular size than CO₂ and therefore is able to penetrate the specimen faster, the pressure gradient would be less than that with the larger molecular size CO₂. The fact that

there is no significant difference between the He and Vacuum test results, indicate that there was no damage occurring in He tests at 3.45 Mpa. However it is conceivable that such damage might be occurring with CO₂, which could explain the lower tensile strength rather than the sorption effect. It was therefore essential to distinguish between these two possibilities.

To be able to make a conclusion, i.e whether the observed difference in strength is caused by adsorption or by the "pressure" effect, it was decided to change the test procedure as described below.

7.2.1 CHANGE IN THE TEST PROCEDURE

Due to the possibility that the application of high CO₂ pressure in a short period might damage the specimen, it was decided that rather than applying the full pressure over a relatively short period, it would be reached in a series of pressure increments. An incremental value of 0.35 Mpa. was chosen because this is well below the strength of coal, thus the damage effect, if exist, should be eliminated because the differential pressure will always be less than or equal to 0.35 Mpa. In addition it was decided to allow the specimen to be saturated for three hours after each pressure increment. This time was considered to be adequate for pressure equilibrium to be attained inside the specimen.

8. MAIN TEST RESULTS AND EVALUATION

8.1 RESULTS

The test results reported in this chapter were obtained with the pressure being applied as described in chapter 7.3.1 He and CO₂ were used as saturation fluids at 345, 1035, and 3450 kPa pressure levels. In addition, the tensile strength of coal was determined at atmospheric pressure. The individual test results are given in Appendices 7,8,9,10, 11, and 12, and are summarised in Table 5.

8.2 EVALUATION OF THE MAIN TEST RESULTS

It is seen that over the total pressure range (0 - 3448 kPa) there is no significant difference between the tensile strengths in air, CO₂, and He (see Appendix 15). The fact that this experimental test procedure shows no significant difference between the He and CO₂ results at 3.45 MPa indicates that the difference observed in the preliminary tests must be attributed to specimen damage during the CO₂ gas injection and cannot be due to adsorption.

8.2.1 CONCLUSION

It is therefore concluded that for this particular coal over the pressure range of 0 - 3.45 MPa,

a) There is no significant change in the tensile strength that can be attributed to change of surface energy due to CO₂ gas sorption. Any change, that might have

occurred but could not be measured due to variations in the normal strength of coal, is found to be less than 5% of the mean strength of coal (see Appendix 15).

b) The differences observed between He and CO₂ at 3.45 Mpa in the preliminary tests have been shown to be attributable to damage caused during the rapid gas injection process. The change of test procedure eliminated this difference.

TABLE 5 : SUMMARY OF THE MAIN TEST RESULTS

GAS	SATURATION PRESSURE (kPa)	NO. OF SAMPLES TESTED	TENSILE STRENGTH	
			MEAN (kPa)	ST.DEV. (kPa)
Air	0	14	978.3	170.4
HE	345	10	946.0	169.7
" "	1035	10	1015.9	218.9
" "	3450	10	999.3	188.6
CO ₂	345	10	988.7	189.7
" "	1035	10	1032.7	360.7
" "	3450	10	960.5	190.9

8.3 DISCUSSION

It was shown by this study that the adsorption of CO_2 gas does not produce a significant reduction on the strength of coal tested. From the review of previous investigations (Chapter 4) it can be seen that there are two studies whose purpose of investigation was parallel to the present study: the study of Ettinger and Lamba (1957) and that of Tankard (1957).

The work of Tankard (1957) was reviewed and analysed in detail in section 4.2. As a result of this analysis it was concluded that there was no strengthening, nor weakening of coal in the gas pressure range of 1 - 20 atm. due to application of the CO_2 gas pressure. There was, however, a significant decrease in the strength of coal when tested under the gas pressures greater than 20 atm. when compared to strength of coal obtained under 1 atm. air pressure. Because the method of experiments required the sample to be taken out for the surface area determinations, the sample could have been damaged, and perhaps subjected to a tensile failure due to release of the gas from the specimen. This damage would have occurred before the intended sorption effect could be measured. Furthermore comparisons were made between results obtained at 1 atm. air pressure and those of CO_2 at various pressure levels. On the other hand, a true comparison requires tests with a non-adsorbing gas (or else air) at the same pressure levels as those of CO_2 . Therefore the reduction in strength, observed by Tankard in the

pressure range of 20 - 30 atm., cannot necessarily be attributable to the adsorption of CO_2 on coal.

Ettinger and Lamba (1957) started with a coal sample comprising 6.5 - 7 mm. size particles. This sample was enclosed in an apparatus, evacuated, and then gassed (CO_2) at 40 atm. pressure. After a 13 hour sorption period, a known weight was dropped on coal 5 times. Then, the gas was let out, the sample re-evacuated, and the crushed coal was transferred to a series of sieves where the amount of coal produced with a particle size less than 0.5 mm. was separated. The weight of this fraction of coal was taken as the hardness index for coal. The hardness index was also determined in air. As it was mentioned in the section 4, these authors concluded that "non disturbed coals with a few cracks" possess equal hardness both in air and in carbon dioxide, while the strength of "disturbed coals" with CO_2 was reduced by a factor of 0.75 when compared to the strength obtained in air, and that the observed reduction was due to sorption effect of CO_2 .

Apart from the obvious difficulties associated with the classification of coal using subjective terms such as "disturbed", "slightly disturbed", and "undisturbed", there are some unclear points that need to be addressed in the work of Ettinger and Lamba (1957) as published:

1. "The hardness of coal was determined in air and in carbon dioxide at a pressure of 40 atm." (Ettinger and Lamba, 1957). This sentence suggests that the hardness

of coal was determined in CO_2 at a pressure of 40 atm, and in air at the same pressure, i.e 40 atm. However, according to Ettinger (1966) the experiments were carried out under 1 atm. air and 40 atm. CO_2 pressures, in which case a true comparison cannot be made, i.e the reduction in strength could be due to the pressure effect as discussed for the work of Tankard (1957).

2. The determination of the hardness index is done after the gas has been released from the samples (there is no mention of the rate of release of the gas). Therefore, the observed difference in strength could equally be due to the pressure gradient created in the sample during the release of gas from the sample. The magnitude of such a pressure gradient will depend on the molecular size of gas and thus on the rate of release of gas from the sample.

Based on these reservations, a positive conclusion, i.e. whether the observed reduction in strength is due to the effect of adsorption or not, cannot be drawn from the work of Ettinger and Lamba (1957) either.

The above criticisms of previous work emphasizes the great care that should be taken in designing, carrying out, and reporting of such experimental work. The following factors must be met if satisfactory and incontrovertible results are to be obtained.

1. Comparison should be made between the strength of coal in sorbing and non-sorbing gases at the same pressure in

order to be sure that there is no "pressure effect" and that any differences observed are due solely to sorption.

2. The rate of gas pressure injection and release should be carefully controlled to ensure that at no stage during the tests, can the gas pressure gradient in the specimen exceed the strength of coal -otherwise the specimen damage may occur- as was the case with the preliminary tests reported in this thesis.

8.4 FURTHER EXPERIMENTATION

8.4.1 TENSILE STRENGTH TESTS, AUSTRALIAN COAL

In addition to the extensive testing on Highvale coal, a total of 11 coal samples from the Metropolitan colliery, Australia were tested for their tensile strength under 500 psi helium and carbon dioxide gas pressures. The results of these tests are listed in Appendix 14, with their summary given below:

PRESSURE/GAS	No. OF SAMPLES TESTED	TENSILE	STRENGTH (kPa)
		MEAN	ST. DEVIATION
3450 kPa/He	5	925.7	115.2
3450 kPa/CO ₂	6	884.8	263.7

According to these results there is no significant difference in the strength of Metropolitan coal resulting from the type of gas being used. Metropolitan coal is of higher rank than Highvale coal, and it is known for its susceptibility to outbursts.

CONCLUSION : Although a limited number of samples were tested at one pressure level only, the tests with Australian coal also indicate that the strength of coal is not affected significantly due to adsorption of CO₂ gas on coal.

8.4.2 DILATION MEASUREMENTS

Gregg (1961, p. 67) referring to the tests on the effect of sorption on the strength of coal compacts (Desai and Gregg, 1958), concludes that "if an ambient vapour is to bring about a weakening of a compacted solid, it is necessary not only that vapour shall be adsorbed on the solid but also that adsorption shall be accompanied by appreciable swelling." In the tests reported in the previous chapters no consideration was given to the influence, if any, of swelling of coal when the gas was adsorbed.

It was therefore decided to conduct some preliminary experiments to estimate the extent of swelling of this coal caused by CO₂ adsorption under the test conditions of the present study.

The apparatus used in the tensile strength tests was used in these experiments with the following attachments:

1. An extensometer with signal amplifier/recorder,

2. A pressure transducer with signal amplifier/recorder,
and
3. An X-Y chart recorder.

The disc sample was secured between extensometer arms via the extensometer chain (Fig. 21), and output wires were taken through the top lid of the cylinder (a leak proof connection was needed) and connected to the X scale input of the chart recorder through the signal amplifier.

The pressure transducer was connected to the gas line and output was transferred to the Y scale input of the chart recorder through another signal amplifier. A schematic drawing of the experiment set up is given in Figure 22.

After the necessary calibrations, the sample, which was left in the cylinder with the extensometer, was subjected to pressure and a graph showing pressure vs. sample expansion was obtained. At each pressure level sufficient time was allowed such that there was no further dilation occurring (no observable pen movement) before increasing the pressure to the next level. Several coal samples were tested in this manner, in addition to a steel disc sample that was tested for comparison.

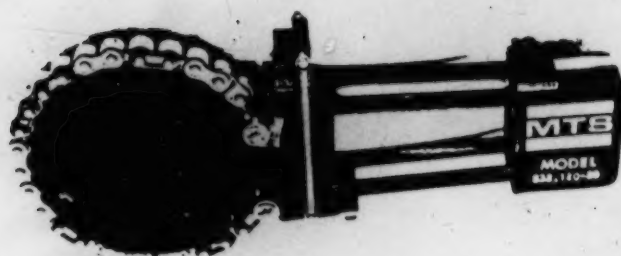


Figure 21: Disc sample with extensometer.

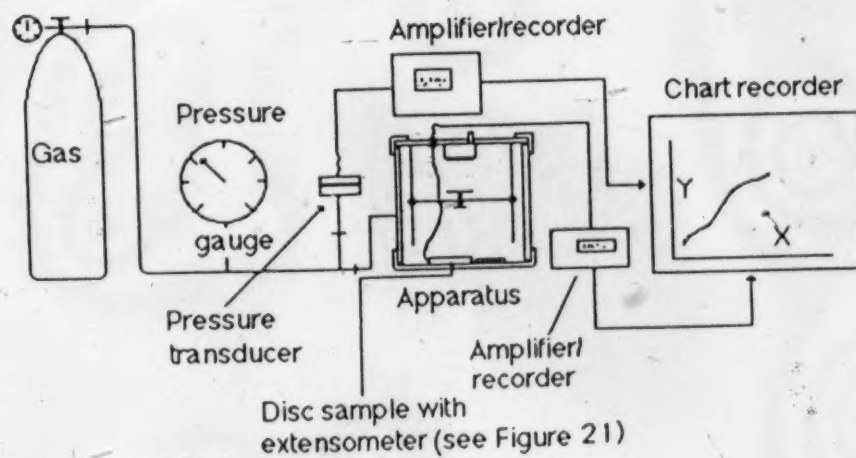


Figure 22: Diagram of dilation experiments.

The typical results obtained from these experiments are tabulated in Table 6, and are plotted in Figure 23 as fractional increase in diameter of the specimen, $\Delta L/L$, vs. the gas pressure. The dilation of the coal with adsorption of CO_2 is readily seen from the results. The magnitude of dilation, 0.57% at 3.45 Mpa, is quite low when compared to those reported by Desai and Gregg (1958) who tested compacted coal specimens with benzene vapour and found 5% dilation, resulting a large decrease in modulus of rupture of coal compacts (see Section 4.1).

When the same coal specimen which has been pressurized and subsequently de-pressurized before was reused in the experiment, the amount of dilation increased from 0.57% to 2.5% at 3.45 Mpa CO_2 gas pressure. This sample was apparently damaged during the desorption stage (although no visual indication of damage was seen), and it can be assumed that more fracture planes were available for adsorption of CO_2 .

8.4.3 CONCLUSIONS

The results from the dilation experiments suggest that:

1. The degree of fracturing is an important factor in the determination of the amount of adsorption, and thus the amount of dilation of coal. However, a fractured sample is already damaged, and thus it is irrelevant to mention the "true" strength of such a sample.
2. Dilation tests may be used for the classification of

coals according to their degree of fracturing. This however requires further investigations.

3. In the light of the item 2 the usage of the coal compacts, that have been subjected to high pressure prior to testing, may not be an adequate method of investigating sorption - strength relationships.

TABLE 6
DILATION TEST RESULTS

GAS PRESSURE (kPa, CO ₂)	$\Delta L/L * 10^{-4}$	
	FRESH SAMPLE	PREVIOUSLY USED SAMPLE
345	4.47	47.90
690	10.77	90.31
1035	17.13	127.20
1380	24.80	152.60
1725	30.49	174.80
2070	36.89	192.05
2415	42.59	209.83
2760	47.72	227.62
3150	52.14	241.65
3450	57.22	254.35

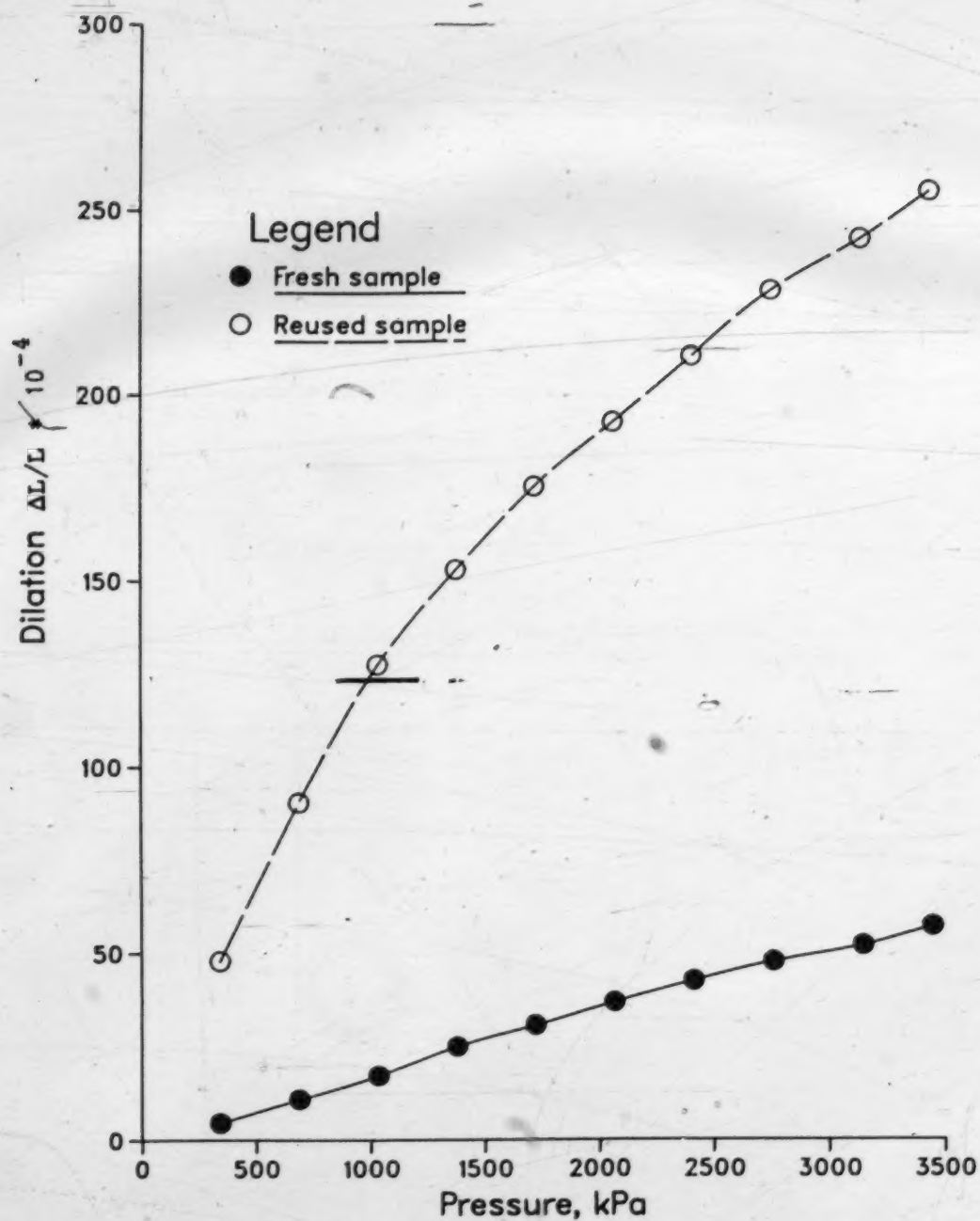


Figure 23: Dilation of disc coal sample vs CO₂ gas pressure.

9. SUMMARY OF CONCLUSIONS AND RECOMMENDATIONS

Under the overall conditions of the experimental work performed in this study, the following conclusions may be drawn:

1. It is possible to measure the tensile strength of solid coal specimens, as opposed to recompactd granular specimens, under gas pressure up to 3.45 Mpa.
 2. By selecting the samples at random, the results can be repeated even with a test material such as coal, whose strength at normal atmospheric pressure is quite variable.
 3. Studies dealing with the effect of sorption on the strength of coal are highly sensitive to the experimental method, and consequently great care must be taken in the designing the experiments to avoid misleading conclusions.
 4. Although theoretically the strength of coal is expected to reduce due to gas sorption, the tests on Highvale coal showed that no significant strength reduction occurred with CO₂ up to 3.45 Mpa gas pressure. For this particular coal and the results, if there were indeed a weakening, but one which could not be detected due to the variation in the normal strength of coal, the magnitude of such a weakening would be below 5% of the mean strength.
- 4a)Preliminary tests with Australian coal taken from a mine where outbursts are known to have occurred, also

indicate a similar result, although insufficient tests were carried out to establish a reliable conclusion for this coal.

5. An analytical outburst model, considering the factors reviewed, is not available and needs to be developed. Such a model, however, will not need to consider a strength reduction due to adsorption of mine gases on coal, if the results from Highvale coal are representative of other rank coals.
6. There is swelling with CO_2 adsorption on coal. However, the magnitude is considerably lower than that reported by Desai and Gregg (1957) with the adsorption of benzene on recompact coal. Because the test materials are completely different, a true comparison could not be made.
7. From the few tests carried out on two different coals, no relationship was found between their ΔP indices and their degree of susceptibility to outbursts.
8. The swelling tests might be used for classification of coal according to the degree of fracturing. However, this will require further investigations.
9. Although no sorption effect was found on the strength of the particular coal used in this study, similar studies are recommended in order to obtain a conclusion for all ranks of coals.

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APPENDIX 1 CLASSIFICATION OF COALS ACCORDING TO RANK^a

Class	Group	Calorific Value MJ/kg ^b	Volatile Matter per cent ^c	Fixed Carbon per cent ^c
Anthracitic	Meta-anthracite		2	98
	Anthracite		8	92
	Semi-anthracite		14	86
Bituminous	Low-volatile bituminous		22	78
	Medium-volatile bituminous		31	69
	High-volatile A bituminous	32.6		
	High-volatile B bituminous	30.2		
	High-volatile C bituminous	26.7		
	Subbituminous Ad	24.4		
Subbituminous	Subbituminous B	22.1		
	Subbituminous C	19.3		
Lignitic	Lignite A	14.7		
	Lignite B			

a Based on ASTM D388-77 in Annual Book of ASTM Standards, Part 26 Gaseous Fuels; Coal and Coke (1980). American Society for Testing and Materials

b Moist, mineral-matter-free basis. (Megajoules per kilogram)

c Dry, mineral-matter-free basis.

d High-volatile C bituminous if agglomerating.

APPENDIX 2 DETERMINATION OF THE INTERNAL SURFACE AREA OF COAL

The determination of the surface area has been a subject of controversy over the years. The intent of this section however, is to present an overview of the methods and their applications in the measurement of the surface area of coal.

There are two methods for the surface area determination:

1. Heat of Immersion Method:

By this method, an evacuated and weighed sample is immersed in a known volume of a suitable liquid, such as methanol. The heat released, ΔH , during this immersion is measured by a calorimeter and is related to the surface area by means of $1 \text{ cal (4.187 J)} = 10 \text{ m}^2$. The surface area found for coal, using this method, was found to be between $20 - 300 \text{ m}^2$, which was considerably higher than that found by using BET method (explained below) from N_2 adsorption. After the findings of Van Krevelen (1954) -that the coal has a fine structure into which N_2 molecules at low temperatures cannot diffuse unless they are provided with a sufficient activation energy- the disagreement in two methods was partly resolved. On the other hand, the heat of immersion method was still being criticized on the basis that

there can be a chemical interaction between the wetting liquid (MeOH) and the coal. This method was therefore abandoned in favour of adsorption (otherwise known as BET) method.

2. Braunauer, Emmet, and Teller (BET) Method:

This method is based on monolayer adsorption of the gases on solid surfaces, and therefore can be seen as an extension of Langmuir's work (Langmuir, 1918). The monolayer capacity is defined as the amount of adsorbate which can be accommodated in a completely filled, single monolayer on the surface of a unit mass (1 g) of the solid. The area of this monolayer can be found by the formulae:

$$S = V_m A_m N_a \dots (1)$$

Where:

S is the total surface covered by the monolayer of the adsorbate, i.e., surface area

V_m is the volume of adsorbate forming the monolayer per gram of the solid, and

A_m is the average area covered by a single molecule of

adsorbate in the completed monolayer ($16.2 \text{ \AA}^2$ for N_2 , and $22.1 \text{ \AA}^2$ for CO_2 are the generally accepted values).

N_a is the Avagadro's number (6.022×10^{23}).

The problem then, is to find V_m . The solution begins by obtaining the adsorption isotherm. The BET equation is then applied to the isotherm data as follows:

The BET equation is:

$$\frac{P}{V(P_0 - P)} = \frac{1}{V_m \beta} + \frac{(\beta - 1)}{V_m \beta} \cdot \frac{P}{P_0}$$

Where:

V is the volume (at NTP) of gas adsorbed at pressure P .

P_0 is the liquefaction pressure of the adsorbate used at the temperature of experiments (at $P = P_0$ the adsorbate condenses to a liquid on the surface of the solid).

β is the constant related to energy of adsorption (the exact value is dependant on the system; it is eliminated in the following procedure).

V_m is the volume of gas required to form a monolayer on

the surface of the solid.

A plot of the left side of the equation against P/P_0 (in the range of 0.05 to 0.35 - presumably the range where the monolayer adsorption occurs) is linear, with

$$\text{intercept } (i) = \frac{1}{V_m \beta} \text{ and}$$

$$\text{slope } (sl) = \frac{\beta - 1}{V_m \beta}.$$

$$\text{by substitution, } V_m = \frac{1}{(sl + i)}$$

The value of V_m , thus found, is then used in equation (1) to determine the surface area.

APPENDIX 3: ENERGY CHANGES ASSOCIATED WITH CRACK PROPAGATION
AND CONDITIONS FOR CRACK INITIATION UNDER PLANE STRESS,
(AFTER HOEK, 1965)

If a crack of length $2c$, in a body subjected to static external tension, is to propagate by a small increment dc , the energy present in the system is redistributed. The changes involved are well explained by Hoek (1965), therefore the following information is quoted verbatim.

1. A loss in potential energy of the externally applied forces due to displacement induced by crack propagation. If the total potential energy of the external forces is denoted by W , the change in W due to changes of dc in crack length is dW/dc .

2. A gain in the excess strain energy in the material around the crack. This excess strain energy is given by the difference between the strain energy in the cracked body and that in the same body without a crack. If the excess strain energy is W_e , the gain in the W_e due to change in dc in crack length is dW_e/dc .

3. A gain in the surface energy of the crack due to the breaking of the molecular bonds along the crack path. If the total surface energy is W_s , then the change in W_s due to a change of dc in crack length is dW_s/dc .
4. A gain in the kinetic energy of the system due to propagation of the crack. If the total kinetic energy in the system is W_k , the increase in W_k due to an increase in the crack length is dW_k/dc .
5. An expenditure of potential energy on plastic deformation at the crack tip in the case of the material which does not exhibit ideal elastic behaviour. The amount of energy dependent on plastic deformation, for an increase of dc in crack length, is dW_{pl}/dc .

Since the energy balance of the system must be maintained,

$$\frac{dW}{dc} = \frac{dW_e}{dc} + \frac{dW_s}{dc} + \frac{dW_k}{dc} + \frac{dW_{pl}}{dc} \dots\dots (1)$$

In case of materials which exhibit considerable plasticity, such as mild steel, the plastic energy term dW_{pl}/dc can dominate the energy balance of the system. On the other

hand, in brittle materials this term is assumed to be negligible compared to the other terms and can be ignored.

Also, for the fracture initiation under static load conditions, it can be assumed that the kinetic energy changes involved are so small that they may be neglected.

Thus, the energy balance governing fracture initiation in a brittle material under static load conditions can be simplified to the following equation:

$$\frac{dW}{dc} = \frac{dW_e}{dc} + \frac{dW_s}{dc} \dots\dots\dots(2)$$

It can be shown that the gain in the excess strain energy around the crack is equal to half the loss in the potential energy of the external forces:

$$\frac{dW_e}{dc} = \frac{dW}{2dc} \dots\dots\dots(3)$$

Hence the equation (2) becomes:

$$\frac{dW_e}{dc} = \frac{dW_s}{dc} \dots\dots\dots(4)$$

In other words, the surface energy gained in rupturing the molecular bonds along the crack path must be equal to the net reduction in the strain energy of the system in order that the crack may propagate.

In order to evaluate equation (4), the excess strain energy in a plain body containing an elliptical crack is calculated. For a thin plate under plane stress conditions

it is found to be:

$$W_e = \frac{c^2 \pi \sigma_a^2}{E} \dots \dots \dots (5)$$

If the plate is thick compared with the dimensions of the crack, then the plain strain conditions exist and the excess strain energy in the system is given by:

$$W_e = \frac{c^2 \pi \sigma_a^2 (1-\nu^2)}{E} \text{ Where } \nu \text{ is the Poisson's ratio of the material.}$$

The surface energy of the crack per unit thickness of plate is given by

$$W_s = 4\gamma c \dots \dots \dots (6)$$

Where

γ is the specific surface energy, i.e the quantity of energy per unit area of the crack surface, $2c$ is the crack length.

According to the equation (4), the condition for fracture initiation is that the change in excess strain energy is equal to the change in surface energy for a change dc in the crack length. Hence, differentiation of equations (5) and (6) with respect to c and solving for applied stress σ_a leads the following conditions for crack initiation under plane stress conditions.

$$\sigma_a = \sqrt{\frac{2\gamma E}{\pi c}} = \sigma_t \dots \dots \dots (7)$$

APPENDIX 4: TENSILE STRENGTH TEST RESULTS (under vacuum)

Experiment No.	Specimen No.	Specimen thickness 10^{-2} m	Specimen breaking pressure (kPa)		Tensile strength (kPa)
			Measured	Corrected	
1	531765	1.346	1151.5	1252.8	703.3
2	600525	1.448	1654.9	1756.2	916.6
3	802145	1.270	1289.4	1390.7	827.5
4	527825	1.245	1275.6	1376.9	836.0
5	710375	1.079	979.2	1080.5	756.4
6	865675	1.021	1068.8	1170.1	866.0
7	187905	1.201	1289.4	1390.7	874.8
8	191165	1.290	1482.5	1583.8	927.6
9	539765	1.041	1310.1	1411.4	1024.2
10	349325	1.232	1296.3	1397.6	857.3

APPENDIX 4 (Cont'd)

11	869825	1.067	1275.6	1376.9	975.4
12	316654	1.240	1310.1	1411.4	860.5
13	736714	1.100	1068.8	1170.1	804.0
14	264234	1.430	2175.3	2276.6	1467.3
15	496474	1.290	999.8	1101.1	644.9
16	681044	1.102	1172.2	1273.5	873.0
17	979164	1.219	1048.1	1149.4	712.4
18	964944	1.097	758.5	859.8	592.1
19	856624	1.189	965.4	1066.7	678.1
20	955244	1.214	1172.2	1273.5	792.7
21	998214	1.079	1282.5	1383.8	968.7

APPENDIX 5: TENSILE STRENGTH TEST RESULTS (PRELIMINARY) UNDER 3450 kPa CO₂ GAS

PRESSURE

Experiment No.	Specimen No.	Specimen thickness 10 ⁻² m	Specimen breaking pressure (kPa)		Tensile strength (kPa)
			Measured	Corrected	
1	594546	1.285	1020.5	1516.9	891.9
2	967726	0.998	413.7	910.1	689.0
3	606936	1.097	482.7	979.1	674.3
4	836626	1.488	1013.6	1510.0	766.6
5	826626	1.171	944.7	1441.1	930.0
6	996656	1.384	903.3	1399.7	764.1
7	572056	1.306	434.4	930.8	538.8
8	938662	1.128	689.5	1185.9	794.7
9	418602	1.130	779.2	1275.6	852.8
10	890282	1.123	427.5	923.9	621.9

APPENDIX 5 (Cont'd)

11	46142	1.204	495.5	992.9	632.2
12	816492	1.229	1068.8	1565.2	962.1
13	759742	1.234	861.9	1358.3	831.5
14	459142	1.227	496.5	992.9	611.6
15	813202	1.097	930.9	1427.3	983.0
16	595122	0.978	737.8	1234.2	953.8
17	593722	1.110	965.3	1461.7	995.2
18	552766	1.161	793.0	1289.4	839.4
19	72646	1.125	875.7	1372.1	921.5
20	211466	1.206	496.5	992.9	621.9

APPENDIX 6: TENSILE STRENGTH TEST RESULTS (PRELIMINARY) UNDER 3450 kPa HELIUM GAS

PRESSURE

Experiment No.	Specimen No.	Specimen thickness 10^{-2} m	Specimen Breaking Pressure (kPa)		Tensile strength (kPa)
			Measured	Corrected	
1	952128	1.232	1310.1	1806.5	1108.2
2	175918	1.156	703.3	1199.7	784.5
3	843718	1.015	496.5	992.9	738.5
4	755898	1.128	1379.0	1875.4	1256.7
5	412558	1.524	1048.1	1544.5	765.8
6	255591	1.262	1034.3	1530.7	916.3
7	953991	1.001	979.1	1475.5	1114.2
8	497901	0.892	634.4	1130.8	958.5
9	295501	1.219	689.5	1185.9	735.1
10	359501	1.130	999.8	1496.2	1000.3
11	58901	1.339	965.3	1461.7	825.2

APPENDIX 7: TENSILE STRENGTH TEST RESULTS, UNDER 345 kPa He GAS PRESSURE

Experiment No.	Specimen No.	Specimen thickness 10^{-2} m	Specimen breaking pressure (kPa)		Tensile Strength (kPa)
			Measured	Corrected	
1	512390	1.100	1413.5	1365.2	938.0
2	488980	1.219	2068.5	2020.2	1252.2
3	104770	1.128	1585.9	1537.6	1030.3
4	871070	1.262	1488.0	1439.7	861.8
5	264090	1.011	1048.1	999.8	747.4
6	518890	1.079	1048.1	999.8	699.9
7	914730	1.135	1551.4	1503.1	1000.4
8	249380	1.097	1310.1	1261.8	869.0
9	705080	1.074	1344.6	1296.3	911.7
10	946310	1.079	1689.3	1641.0	1148.8

APPENDIX 8: UNIAXIAL TENSILE STRENGTH TESTS, UNDER 345 kPa CO₂ GAS PRESSURE

Experiment No.	Specimen No.	Specimen thickness 10 ⁻² m	Specimen breaking pressure (kPa)		Tensile strength (kPa)
			Measured	Corrected	
1	981363	1.191	1392.8	1344.5	852.9
2	643363	1.072	1310.1	1261.8	890.1
3	379763	1.209	1827.2	1778.9	1111.9
4	986673	1.184	1758.2	1703.1	1087.3
5	132653	1.194	1034.3	986.0	624.1
6	108407	1.242	1792.7	1744.4	1061.3
7	458257	1.333	2378.8	2330.5	1320.7
8	983847	1.206	1448.0	1399.7	876.7
9	672427	1.199	1827.2	1723.8	1086.5
10	448890	1.041	1392.8	1344.5	975.7

APPENDIX 9: UNIAXIAL TENSILE STRENGTH TESTS, UNDER 1035 kPa He GAS PRESSURE

Experiment No.	Specimen No.	Specimen thickness 10^{-2} m	Specimen breaking pressure (kPa)		Tensile strength (kPa)
			Measured	Corrected	
1	670181	1.240	1978.9	2054.7	1252.7
2	321591	0.955	806.8	882.6	698.3
3	110867	1.293	1965.1	2040.9	1192.9
4	770181	1.247	1103.2	1179.0	714.4
5	553783	1.250	1999.6	2075.4	1255.0
6	765688	1.270	1241.1	1316.9	783.6
7	722297	1.128	1737.6	1813.4	1215.1
8	954660	1.430	1792.7	1868.5	987.4
9	900601	1.549	2034.1	2109.9	1029.1
10	110907	1.219	1585.9	1661.7	1030.0

APPENDIX 10 UNIAXIAL TENSILE STRENGTH TESTS, UNDER 1035 kPa CO₂ GAS PRESSURE

Experiment No.	Specimen No.	Specimen thickness 10^{-2} m	Specimen breaking pressure (kPa)		Tensile strength (kPa)
			Measured	Corrected	
1	322704	1.097	1585.8	1661.7	1144.4
2	619514	1.499	3392.4	3468.2	1748.9
3	735237	1.270	1723.8	1799.6	1070.8
4	127249	1.085	1999.5	2075.4	1446.1
5	547699	1.618	2206.4	2282.2	1065.9
6	132761	1.227	1482.4	1558.3	959.9
7	528111	1.138	1310.1	1385.9	920.4
8	8201	1.478	930.8	1006.7	514.6
9	155371	1.346	1241.1	1316.9	739.3
10	328341	1.410	1261.8	1337.6	717.1

APPENDIX 11 UNIAXIAL TENSILE STRENGTH TESTS, UNDER 3450 kPa He GAS PRESSURE

Experiment No.	Specimen No.	Specimen thickness 10^{-2} m	Specimen breaking pressure (kPa)		Tensile strength (kPa)
			Measured	Corrected	
1	45301	1.262	1172.2	1668.6	998.9
2	474492	0.952	689.5	1185.9	940.9
3	352682	1.021	951.5	1447.9	1071.6
4	648682	1.587	792.9	1289.4	613.8
5	132537	1.173	1137.7	1634.1	1053.6
6	294532	1.079	1275.6	1772.0	1240.5
7	95870	1.008	586.1	1082.5	811.3
8	362888	1.270	1585.9	2082.3	1239.0
9	663700	1.003	758.5	1254.9	945.2
10	445163	1.097	1068.7	1565.2	1077.9

APPENDIX 12: UNIAXIAL TENSILE STRENGTH TESTS, UNDER 3450 kPa CO₂ GAS PRESSURE

Experiment No.	Specimen No.	Specimen thickness 10 ⁻² m	Specimen breaking pressure (kPa)		Tensile strength (kPa)
			Measured	Corrected	
1	468027	1.209	965.3	1461.7	913.6
2	931301	1.085	827.4	1323.8	922.4
3	869811	1.003	827.4	1323.8	997.1
4	226789	1.410	1172.2	1668.6	894.5
5	163737	1.351	1379.0	1875.4	1048.8
6	962800	1.397	2068.5	2564.9	1387.5
7	2/65481	1.262	1137.7	1634.1	978.2
8	2/42481	1.143	1068.7	1565.2	1034.8
9	2/83452	1.148	551.6	1048.0	689.8
10	2/37542	1.250	724.0	1220.4	738.0

**APPENDIX 13: UNIAXIAL TENSILE STRENGTH RESULTS, UNDER
ATMOSPHERIC PRESSURE**

Experiment No.	Specimen No.	Specimen thickness 10^{-2} m	Specimen breaking pressure (kPa)	Tensile strength (kPa)
1	629161	1.118	1551.4	1049.0
2	221281	0.942	758.4	608.2
3	726439	1.156	1241.1	811.5
4	346669	1.242	1654.8	1006.8
5	831429	1.224	1530.7	944.8
6	774729	1.214	1461.7	909.8
7	673029	1.293	1620.3	947.1
8	537259	1.143	1654.8	1094.1
9	621419	1.130	1896.1	1267.7
10	883519	1.102	1447.9	992.6
11	525079	1.166	1916.8	1242.4
12	544189	1.224	1323.8	817.1
13	898269	1.100	1379.0	947.5
14	934389	1.405	1965.1	1057.2

**APPENDIX 14: TENSILE STRENGTH TEST RESULTS UNDER 3450 kPa,
FOR AUSTRALIAN COAL**

Experiment No.	Specimen thickness 10^{-2} m	Specimen breaking pressure (kPa) (Measured)	Specimen breaking pressure (kPa) (Corrected)	Tensile strength (kPa)
HELIUM				
1	1.227	827.4	1323.8	815.5
2	1.184	862.0	1358.3	867.2
3	1.181	827.4	1323.8	847.0
4	1.224	1172.2	1668.6	1029.9
5	1.326	1379.0	1875.4	1068.9
CO₂				
1	1.085	620.6	1117.0	778.3
2	1.384	275.8	772.2	421.6
3	1.087	896.4	1392.8	968.2
4	0.980	689.5	1185.9	914.2
5	1.199	1379.0	1875.4	1182.1
6	1.107	1034.3	1530.7	1044.5

APPENDIX 15: STATISTICAL COMPARISONS

The Student's t-test is the statistical decision method used in this study for comparisons of two groups of results. Each group has a number of independent individual results (tensile strength in this case) obtained by testing samples, randomly selected from the total population of samples. The decision, i.e. whether two groups of results are different from each other, requires the statement of a hypothesis and a level of significance at which the hypothesis is to be tested.

Hypothesis: Suppose we want to compare the tensile strength results obtained by testing coal with He (sample mean x_1) and CO₂ (sample mean x_2) both under 3450 kPa (500 psi) gas pressure. We state hypothesis as:

H_0 (Null Hypothesis) : $x_1 = \mu = x_2$.

i.e, there is no difference between the means of two results; any observed differences are merely due to fluctuations in sampling from the *same* population, and test this against

H_1 (Alternative Hypothesis): $x_1 = \mu_1 \neq x_2$,

i.e the results of two groups are different from each other.

Type I and type II errors, level of significance:

By formulating the problem with the above hypothesis, we can make two types of error:

Type I error : If we reject a hypothesis when it should be accepted, and

Type II error : If we accept a hypothesis when it should be rejected.

In testing a given hypothesis, the maximum probability with which we would be willing to risk a type I error is called the level of significance of the test and it is decided before the experiments are actually performed. In this study a level of significance of 5% is used, which means that there are about 5 chances in 100 that a hypothesis was rejected when it should have been accepted, i.e we are about 95% confident that we have made the right decision. Once the type I error is chosen, it is then sought to minimize the type II error which usually requires increasing the number of samples.

Example calculation: Two groups of the tensile strength results, obtained by testing the Highvale coal with He and CO₂ at 3450 kPa, summarised below. It is required to know if the mean strength as obtained with one type of gas is significantly different from the mean strength obtained with the other gas.

He

No. of samples tested (n_1) = 10

Sample mean tensile strength ($\bar{x}_1$) = 999.3 kPa.

sample standard deviation of mean (s_1) = 188.6 kPa.

CO₂

No. of samples tested (n_2) = 10

Sample mean tensile strength ($\bar{x}_2$) = 960.5 kPa.

Sample standard deviation of the mean (s_2) = 190.9

kPa.

The null hypothesis, that the sample mean values x_1 and x_2 differ only randomly with an error probability of 0.05, is accepted if the calculated value t_c for the test variables,

$$t = \frac{x_1 - x_2}{s_d} \sqrt{\left(\frac{n_1 n_2}{n_1 + n_2}\right)}, \text{ with}$$

$$s_d^2 = \frac{s_1^2(n_1 - 1) + s_2^2(n_2 - 1)}{n_1 + n_2 - 2}$$

is less than the t_T for error probability 0.05 and $f = (n_1 + n_2 - 2)$ degrees of freedom. Thus,

$$s_d^2 = \frac{35569.96 \cdot 9 + 36442.81 \cdot 9}{18} = 36006.39$$

$$s_d = 189.75, \text{ and}$$

$$t_c = \frac{999.3 - 960.5}{189.75} \sqrt{\frac{100}{20}} = 0.46$$

From the table for the t test, for error probability of 0.05 and $fod = 18$ degrees of freedom, $t_T = 2.10$. Since $t_c < t_T$, the null hypothesis must be accepted. This means that there is no significant difference between the two means.

If the results of more than two groups are to be compared then, the analysis of variance is required. The analysis of variance procedure, by which the comparison can be made of more than two groups of data for equality, is an

extension of the t-test. The hypothesis and alternate, in this case, are:

$$H_0 : \mu_1 = \mu_2 = \dots = \mu_r$$

H_1 : At least two means are not equal.

Where μ_j , ($j = 1, 2, \dots, r$) is the mean of j th group.

The comparison is made on the basis of F_C calculated from the experiment results and F_T found from the statistical tables. If it was found that $F_T < F_C$, then the equal means hypothesis is rejected. Otherwise it is concluded that there is no significant difference in means at the specified significance level.

The following table summarises the comparisons made between the tensile strength results obtained with two gases (He and CO₂) at three pressure levels (345, 1035, and 3450 kPa)

ANALYSIS OF VARIANCE TABLE

SOURCE	SUM OF SQUARES	D.o.f.	MEAN SQUARE	F VALUE
GAS TYPE	734.1	1	734.1	0.01
PRESSURE LEVEL	35884.9	2	17942.5	0.34
INTERACTIONS	17342.9	2	8671.4	0.17
ERROR	2833319.4	54	52468.9	

From the statistical tables, for error probability of 0.05, the F value is found to be greater than those found from the analysis of variance:

F_T , gas type : 4.0

F_T , pressure level: 3.1

F_T , interactions : 3.1

Therefore it can be concluded that neither the gas type nor the pressure level produces a significant difference in the mean tensile strength of the Highvale coal.

Because it was shown that there is no significant difference between the means of all the final test results, they can be combined to obtain the population mean and the standard deviation of the mean. Thus,

Total number of samples (main tests) = 74

Mean tensile strength = 989.4 kPa.

Standard deviation of the mean = 212.4 kPa.

The 95% confidence interval is :

$$x \pm z_{\alpha/2} \frac{s}{\sqrt{n}}$$

$$x \pm z_{0.025} \frac{212.4}{\sqrt{74}} = x \pm 1.98 \frac{212.4}{\sqrt{74}} = x \pm 48.9 \text{ kPa}$$

$$940.5 < x < 1038.3$$

Thus in case attempts are made to repeat 74 tests under any of the test condition mentioned, the results will come out to be in this interval 95 times out of 100 attempts, and will not differ significantly from each other. It can also be concluded that if the strengthening or weakening of this particular coal occurred due to the type of gas used in the experiments, but was masked due to variation in the normal strength of coal, the amount of this weakening (or strengthening) would be less than 5% of the mean strength of the combined results.